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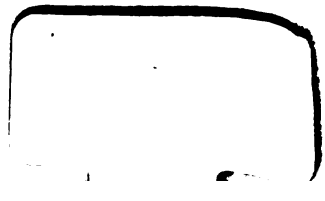

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SOME PHASES OF THE AMERICAN RAILROAD PROBLEM.*

BY STUYVESANT FISH, A.M.

In the United States, as in Great Britain, railroads have been built and are maintained through the use of private capital. With us railroad companies are corporations, and therefore citizens of the several States, while in Great Britain they have invariably been chartered by Parliament. The older charters granted by our States are contracts, irrevocable by them.

Long prior to the introduction of transportation by steam on rails there had been incorporated in Great Britain and in the United States turnpike companies and canal companies. In every case the corporation was created for the purpose of producing, at the cost and risk of private capital, a public highway free to all on equal terms under like circumstances, upon the payment of toll, such toll to be determined by the corporation within limits fixed in its charter. On turnpikes and canals alike the motive power was that of horses or mules. As the corporation did not furnish the motive power, and as the capacity of such beasts of burden was known to be much greater in respect to freight in canal boats than in wagons, that fact was recognized in the acts of incorporation, which, in the United States, at least, generally allowed a toll of about one cent (1c.) per ton per mile on the turnpikes, and of about six and one-quarter cents (6¼c.) per ton per mile on the canals. These were, however, merely tolls for the use of the highway, and did not involve on the part of the corporation any expense or risk in respect to providing the vehicle, moving it or its contents, safe-

* Address delivered before the graduating class in Applied Science, Columbia University, Monday, June 10, 1907.

guarding or insuring the persons or goods transported, or guaranteeing their safe and speedy delivery. The turnpike or canal company had no concern with such matters, all of which were at the added cost and at the sole risk of the carrier or the shipper. This distinction between the toll exacted for the use of the highway and the carriers' charge, which is a very different, and, of necessity, a much greater item, must constantly be borne in mind. It is probably due to the fact of the preëxistence of such toll roads and canals, built, maintained and operated by private capital, that in these two English-speaking countries, railroads have thus far been similarly built, maintained and operated by private corporations at their sole risk and for their individual profit.

In the United States, excepting a small part of the more densely populated districts immediately adjoining the Atlantic Seaboard and chiefly in the Northern States, the railways were built in advance of civilization, and have, as foreign writers have remarked more frequently than our own, created our civilization as well as the traffic which they now carry. In Great Britain, on the other hand, railroads were not undertaken until long after the establishment of towns, markets and seaports on the sites now occupied by them, and until centuries after good roads had been built between most of them at the public cost.

The genesis of railroads took place in this country and in Great Britain at about the same time, say in 1830. Their construction was stimulated and made commercially profitable, and therefore possible, through the invention and gradual perfection of the steam locomotive engine. There was nothing new at that time in the use of rails, whether of wood or of iron; for the fact that through thus providing a smooth, firm and level surface, a draft animal could haul much greater loads than over any form of road, however good, had been known for centuries; and rails of wood, and later of iron, had been extensively used, especially in and about the mines. The term "railroads" is therefore somewhat of a misnomer, they owing their existence to the application of steam to locomotion on land, rather than to the use of rails. We will this year celebrate here in New York the one hundredth anniversary of the navigation of the waters of the Hudson River by Robert Fulton's *Claremont*. His steamboat was by no means the first of its kind, nor even the first which he had built and successfully moved; but as he himself stated, it was the first to prove a permanent commercial success.

Another American engineer, Oliver Evans, had in 1804 constructed in Philadelphia a machine with the strange title "Eructor Amphibolos," which after moving a mile and a half over the streets of that city, was driven on to a boat, and there through the means of a wheel at the stern of the boat, provided the power for moving it and its cargo for a distance of sixteen miles on the Schuylkill River. To Evans is also due the far greater credit of having been the first in this country or abroad to invent and patent a high pressure steam engine, his patent granted by the State of Maryland May 21, 1787, having antedated the English patent of Trevethick and Vivian which was issued in March, 1802, by nearly fifteen years. Indeed Evans' influence in that direction, as opposed to that of Watts in England, who was and remained the advocate of low pressure engines, resulted in the speedy development of steamboat service on our rivers and lakes, after Fulton had shown it to be commercially profitable, and to the general adoption in the United States of the high pressure engine, without which things could not have gone on with us as rapidly as they did. While on this subject, it may not be amiss to also recall the fact that John Stevens, whose name you will find in the records of Columbia University as having graduated in the class of 1768, built a steamboat, *The Phœnix*, which in the year 1808 was the first to navigate the high seas; at least so far as to run from New York to Philadelphia. To Stevens is also attributed the credit of having built the first American locomotive.

The British railways found an existing traffic, supported by a dense population, awaiting their activities, and had in their own country every facility for producing all the rails, cars, engines and other equipment needed, together with an abundance of coal for fuel, and capital seeking investment at reasonably low rates. They were, however, confronted by enormous outlays for the purchase of property running through cultivated lands and inhabited towns and cities, and by the necessity at the outset of separating the plane of traffic by rail from that of vehicles, their railways being carried over or under all existing roads and streets. This resulted in the original capital cost of English railways vastly exceeding that of ours.

The American railways, on the other hand, were built through a sparsely settled country, and often through an uninhabited desert, and were the means of introducing population and civilization, and

thereby creating traffic. So destitute was this country of facilities for producing rails, that the Baltimore & Ohio Railroad Company, the Delaware & Hudson Canal Company and the South Carolina Railroad Company, each separately memorialized Congress to the effect that the rolling mills of the United States were not competent to produce the amount of iron needed for rails alone, within the time which had been set by their several charters for the completion of their railroads, and that it would therefore be necessary to import the rails from Europe, and requested a remission of the duty thereon. Congress impressed by this passed an "Act to release from duty iron prepared for and actually laid on railways or inclined planes," which was approved by President Jackson on July 14, 1832, and continued in effect for eleven years and until 1843. In like manner our earliest locomotive engines were imported from England to which fact is doubtless due the similarity of our railroad gauge to theirs. Capital was almost unobtainable, and what little could be secured was borrowed at extortionate rates. The statutes of almost every State contained provisions authorizing railroad companies to borrow at rates far above those which in all other transactions were forbidden as usurious. Every sort of device had to be resorted to, to eke out the meager capital thus secured, and cities of the first class did not hesitate to authorize railroad companies to lay down and operate by steam, tracks in and across their most densely traveled thoroughfares. With us it was a question of how, and at the least possible cost, to lay down a set of rails over which a steam engine could be moved. Such questions as the speed of trains, the economy of transportation, and even the safety of passengers, and of the public generally, had to be ignored. Nor were these the conditions with us at the outset only, but they prevailed very largely until comparatively recent times, so that it is only now that the American railroads, even in the more densely populated parts of the country, find themselves face to face with the problem of effectually and absolutely separating the plane of railroad traffic from that of street traffic.

Some of the States on the Atlantic Seaboard and many of those in the Middle West had, before the introduction of railroads, undertaken to build canals at the public cost. The general experience with this form of government ownership of public utilities was not satisfactory. Indiana and Illinois which, in 1827, had received the

first congressional grant of lands in aid of any works of public utility — in their case canals — had fared badly; so also Ohio, Pennsylvania and other States. The work had been attended with great scandal, much maladministration, more politics, and had not infrequently been followed by defaults on the State debts.

From an interesting paper on "Railway Regulation under Foreign and Domestic Laws," prepared under the direction of the Federal Industrial Commission by Professor B. H. Meyer, Ph.D., of the University of Wisconsin, we learn that the early railroad charters in some cases provide that the company may charge "such rates per mile as may be agreed upon and established from time to time by the directors of said corporation." Some States went so far as to empower the company to "change, lower, or raise rates at pleasure, provided that the rates established from time to time shall be posted in some conspicuous place or places."

The early charters often contained provisions making a distinct difference between the toll to be charged by the railroad company as the owner of the highway, for allowing others the privilege of moving their own goods in their own vehicles over such highway, and the higher charge to be made by the company for transportation when it furnished in addition to the highway, the vehicle and the means of moving it.

In 1829 Connecticut specifically authorized the following rates per ton per mile,— for sixty miles five cents, for between twenty and sixty miles six cents, for less than twenty miles seven cents, thereby recognizing the relatively higher cost of handling short haul traffic.

In 1837 North Carolina granted a charter which provided for maximum rates as follows: "On persons, not exceeding 6 cents per mile for each, unless the distances to which any person be transported be less than 10 miles, in which case the president and board of directors may be entitled to make an extra charge of 50 cents for taking up and putting down each person so transported"; thus recognizing the value of and providing an added charge for special and local service.

A very common inducement for the investment of capital in railroads was through the giving of exemptions from taxation. Banking privileges were not infrequently given to railroad companies.

As showing the charges actually levied, Mr. E. A. Moseley, the Secretary of the Interstate Commerce Commission, is my authority

for the statement that—"In 1837 the Superintendent of Motive Power of the Columbia & Philadelphia Railroad reported that the following charges were imposed on the railroads named:

FREIGHT RATES OF RAILROADS IN 1837.

Railroad.	Per Ton Per Mile. Cents.
Baltimore & Ohio	4½
Baltimore & Washington	4
Winchester & Potomac	7
Portsmouth & Roanoke	8
Boston & Providence	10
Boston & Lowell	7
Mohawk & Hudson	8
Petersburg	10"

Under the heading of "State Participation," Professor Meyer truly says: "The history of internal improvements had been such as to discourage the active participation of our commonwealths in the construction of railways. Works of internal improvement, greatly exceeding both the capacity to construct and to utilize them, had been projected by many States. The inevitable failure of these gigantic projects brought these States into disrepute as active economic agents; hence we find in constitutions and charters granted after this period of disaster in State works of internal improvement, direct prohibitions of State participation."

Our States, following the example of Great Britain, wisely determined to have railroads built by the private capital of corporations created for that purpose, and it was due to this fact that as early as 1850 the United States contained as many miles of railroad as all Europe.

The history of American railroads may be divided into five periods:

I. 1830–1 50. EARLY LEGISLATION.

A disposition on the part of the several States to grant most liberal and subsequently still broader powers, and even to invest public moneys and pledge the public credit in aid of railroads owned and controlled by private corporations. This continued until about 1850, and in many States much louger.

II. 1850–1870. LAND GRANTS.

The lack of capital needed for building railroads, and the impossibility of selling even the fertile lands of the Middle West and

South which were not accessible to navigable waters, led to the passage by Congress of grants of public lands to the States of Illinois, Mississippi and Alabama in aid of railroads to be built from Chicago to Mobile. This was followed by many similar grants to various States in aid of railroads to be built within their boundaries. Shortly after the breaking out of the Civil War the same considerations, with the added ones of providing for military purposes and of tying California to the Union, led to the passage of acts granting vast amounts of land, as well as millions of government bonds, in aid of the construction of a railroad through the Territories from the Missouri River to California. The grants in these cases were made directly to corporations chartered by the Federal Government, and from that time on many federal grants of land and issues of Government bonds were made directly to corporations. That in view of the circumstances then prevailing the judgment displayed by our national legislators was sound, is proved by the growth of our western territory, and by the fact that the Government has already been reimbursed for nearly all, and will be reimbursed for all, the advances made in the shape of bonds. It goes without saying that the grant of inaccessible and unsalable alternate sections of wild lands, reserving to the Government the other sections, was wise, in that the reserved sections thereby became salable at fair prices. And this is true, notwithstanding the many scandals which attended the construction of some of those railroads and the handling of their land grants.

III. 1870–1887. Granger Legislation.

After the close of the Civil War, the so-called "Granger Legislation" began to be enacted by the Western States, through the people of the West asserting their rights as against the extortionate rates and the unjust discriminations then prevailing on certain railroads. This contest went on with varying success in the courts and in the legislatures, until in 1887 the courts having held that a State could not fix or limit rates of transportation from a point in one State to a point in another, Congress, through the Act to Regulate Commerce, took jurisdiction of common carriers engaged in transportation from points in one State to points in another. The Act of 1887, however, specifically provides that it shall not apply to transportation "wholly within one State." That is to say, Congress then assumed in respect to railroads but a part of

the power delegated to it by the Constitution. For the provision of the Constitution is that Congress shall have power "To regulate Commerce with foreign nations, *and among the several States*, and with the Indian Tribes."

I thought then and I think now, that if at any time it should become desirable for the Federal Government to take over the regulation of all commerce among the several States, including that which has both origin and destination within the bounds of the same State, it would not be difficult to amend the law of 1887 by striking out the limitations upon the jurisdiction of the commission thereby created, and by subjecting to the new law all railroads engaged in transportation among the several States, without regard to whether such transportation has origin and destination within the same State. I am not a lawyer, much less a constitutional lawyer, and cannot therefore, say whether such a law would be held constitutional by the Supreme Court, but I do know that the word "among" has a different and a wider meaning than the word "between;" and if a thing rises to the dignity of "commerce," as distinguished from "petty trade," I cannot but believe that the wise and experienced men who framed the Constitution meant precisely what they said—that Congress should have power to regulate it "among the several States," and not merely between them. With all submission, it also seems to me that our worthy President, Mr. Roosevelt, is more likely to find a means of bringing about what he seems to desire through this clause of the Constitution than through that relating to post routes.

IV. 1887-1906. Interstate Commerce Commission.

The body created by the Act of 1887 to Regulate Commerce was called the "Interstate Commerce Commission." To it were given powers in relation to the exaction of reports from railroad companies authorizing it to go into the most minute details of what I at least conceive to be the private business of corporations formed under the laws of the several States for gain, and to that part of it which relate solely to their traffic within a particular State. Without going into a discussion of the authority of Congress to exact such reports, or the wisdom of so doing, this has resulted in one thing of inestimable good to the public, and especially to the shareholders and creditors of railroad corporations, to wit, a very substantial uniformity in railroad reports, and, what is

more important, their publication. No one who was not familiar with railroad reports prior to 1887, and indeed some years thereafter, can now realize what misleading and imperfect documents many—I might say most of them—were.

V. The Present and the Future.

In 1906 there was given to the Interstate Commerce Commission, at it urgent solicitation, the added power of fixing rates. While it is true this power is only to be exercised after investigation, and subject to review by the courts, this constitutes a distinct reversion to the medieval practice of attempting by statute to fix the price of commodities and services. All the nations of Europe had a long experience with laws intended to fix, as between the buyer and the seller, what the priestly lawgivers of the Dark Ages called "Justum Pretium." While their laws did not forbid the buyer and the seller from trafficking together, they did attempt to say at what prices things should be bought and sold. After centuries of such vain effort to establish the just price,—for a loaf of bread, for the wages of servants, the rent of land, the rate of interest, and many other things, the attempt was very generally abandoned, before the adoption of our present form of government, since which time prices have been left to the free play of the law of supply and demand. It is to my mind amazing that the Congress of the United States should with practical unanimity have made this distinct step backward; for it will certainly hinder and delay, if it does not in some places absolutely prevent, the development of the West and South through the establishment of new jobbing centers.

It would also seem that under the new law we are to have a valuation of the railroads for the purpose of ascertaining whether or not any of them, and, if so, which, are over-capitalized — and this apparently to the end of fixing just rates for railroad freights and fares in proportion to a true valuation. If such valuation were possible, which I doubt, and the fixing *in advance* by any governmental agency of a just rate were possible, which I deny emphatically, the result will probably be that no railroads will be built in the undeveloped parts of this country, except at the risk and with the capital of existing lines, for no man would venture his capital in the construction of a railroad in a new country for anything like the five or six per cent. per annum which might be allowed by government as a fair return thereon. In this respect we are likely

to have from the law of 1906 precisely the same experience as was had from that part of the law of 1887 forbidding pooling; from which it has resulted that since 1887 no railroads have been built for a share in pools, and the only builders of railroads of any magnitude during the last twenty years have been, directly or indirectly, the large systems then existing or since brought together.

As one experienced in the business and interested pecuniarily in railroad shares, I can and do look upon what is going on under the law of 1906 with equanimity, much as I deprecate the unwisdom of that law as a citizen having regard to the welfare of the whole of our common country.

In saying this, let me not be understood as opposed to the regulation of railroads by the state or by the federal authorities, or by both, for in common with nearly all railroad men I appreciate the justice, the wisdom and the necessity of such control. What I do object to is the attempt to fix, in advance by statute, the price of railroad rates or any other service or commodity.

Capitalization.

So much has been said about the alleged over-capitalization of railroads, and so many glaring instances of rascalities in respect thereto have been brought to light, some of them even in recent years, that the public have come to believe that the railroads as a whole are over-capitalized. I am free to admit that prior to 1880 the American railroads taken as a whole were over-capitalized in the sense that the sum total of their bonds and shares then outstanding exceeded the value, although it did not then exceed the cost of their property. My meaning can be explained by an instance right at our doors. When I was a student at Columbia College, the New York Central & Hudson River Railroad ran its trains at the level of the streets in Fourth Avenue at the back of the College. About the time of my graduation the grade through Harlem was lowered and a new bridge built over the river. Some years since we saw all that work thrown away, and the tracks elevated to their present grade, with a new bridge over the Harlem River, and we now see the same company spending many millions in making an enormous excavation on the same land back of the old College in Fourth avenue near Forty-ninth Street. All this work of construction, change, demolition and reconstruction has added enormously to the cost without adding in like degree to the

value of the property as it now stands. Hence the difference which I make between "cost" and "value" in 1880. Since that time, however, railways as a whole have gone through an era of tremendous under-capitalization, through the application of net income to betterments, and through reorganizations, which were forced on many of them in the bad years following 1893, when for a long period fully one-quarter of the railway mileage in the United States was in the hands of receivers. Bonds stipulating for the payment of high rates of interest, six, seven and eight per cent., and secured by mortgages, were funded into other bonds, often for less amounts, and always at a much lower rate of interest.

The latest report of the Interstate Commerce Commission on Statistics of Railways in the United States is that for the year ended June 30, 1905. It states (pp. 56–57) that the total of stocks and bonds, not owned by railroad companies was:

In 1890	$7,126,673,041
In 1905	9,940,853,945
This shows an increase of	$2,814,180,904

or less than four-tenths, to be exact 39.49 per cent., in the total of stocks and bonds outstanding in the hands of the general public.

In those same fifteen years the number of miles of line has been increased in almost exactly the same ratio, to wit, 38.73 per cent., but the mileage of second, third and fourth main tracks has been more than double (increase 103.70 per cent.), and that of yard tracks, sidings and spurs has been increased by over four-fifths (83.74 per cent.) while the length of all tracks has increased by more than one-half (53.49 per cent.). The number of engines has been increased by nearly one-half (49.50 per cent.), and that of cars of all kinds by more (50.16 per cent.). Grades and curves have been reduced, stone and iron bridges substituted for wooden structures, the average weight of rails has been increased by fully one-quarter and the capacity of engines and cars in a much greater ratio. Interlocking devices, air brakes, automatic couplers and other safety devices have been added, and terminals vastly enlarged.

The true way to look at this question, however, is in regard to the service rendered, for the value of a machine (and a railroad is but little more) grows very nearly in proportion to the volume of its output. Our railroads carried:

Of passengers one mile,

In 1890	11,847,785,617	
And in 1905	23,800,149,436	
Increase	11,952,363,819	or 100.88 per cent.

Of tons of freight one mile,

In 1890	76,207,047,298	
And in 1905	186,463,109,510	
Increase	110,256,062,212	or 144.68 per cent.

And adding these two somewhat dissimilar items together we find that the railroads rendered to the public:

Of units of service,

In 1890	88,054,832,915	
And in 1905	210,263,258,946	
Increase	122,208,246,031	or 138.79 per cent.

I submit that apart from the great improvements in their tangible property shown above, our railroads which are now rendering nearly two and a half times as much service as they did fifteen years ago, and better, far cheaper and quicker service at that, are intrinsically worth vastly in excess of the forty per cent. increase shown in their outstanding stocks and bonds.

It was found impossible to build railroads as rapidly as the growth of the country called for them, solely, or even largely, with a capital paid in cash by the shareholders. Hence of necessity bonds were often sold in advance of construction, nor could they be sold except at a discount and with a bonus of stock thrown in. In this there was the same over-capitalization as occurs daily when a merchant needing capital in his business sells his notes at a discount or subject to large commissions.

Against this yielding to the conditions confronting the problem I cannot conceive that any one will object. But unfortunately other things were done right here in New York and on a vast scale, in the fraudulent over-issue of stocks, in the secret and sudden conversion of scrip into stock, and in juggling with dividends and with accounts. Bad as all this was it did not permeate the whole railroad system and there were even then scores of companies which kept the faith and steadily improved their properties out of current earnings. And after all, these frauds bear more heavily on the stockholders and creditors of the companies than on the public.

From the day I graduated from Columbia College to this, I have been continuously interested in and for most of that time employed by railroad companies, and have read somewhat extensively on the subject. As a result of such experience I have no hesitancy in saying that no railroad fortune was ever made through enhancing rates, oppressing shippers, or withstanding the general tendency of rates to decrease. And what is more, every dishonest railroad fortune has been made, not by oppressing the shippers, but through robbing the stockholders. Should you ask why these stockholders have not sued for restitution, I would remind you of the cost and delay of such litigation, and of the fact that if restitution should be made, it would be to the corporation, of which in all probability the same persons would remain in control, as the majority holders and as officers and directors, so that the funds restored would simply revert to their custody and their tender mercy.

The Valuation of Railroads.

Much has been said of late of a proposed valuation of the railroads, although it is admitted on all hands that the capitalization of railroads has, and can have, no bearing whatever on the rates charged. Moreover in whatever manner a valuation may be made, it must in greater or less degree be based on earning power, which in turn rests on the rates charged. Hence the predication of rates on a valuation would merely be working around in a circle.

There are vast differences in the capitalization of the four great Trunk Lines between Chicago and New York, namely, the Pennsylvania, the New York Central, the Erie and the Baltimore & Ohio. The first two have paid dividends steadily for more than twenty years; the Baltimore & Ohio has been reorganized at least once, and the Erie more than once in that period, and has not paid a dollar of dividends on its common stock, and yet the rates of freight charged on all four to common points have been and are identical.

Such valuation, while costly and of necessity inquisitorial, can result in little but a redemonstration of the fact that our railroads are under-capitalized, for the Bureau of the Census has already computed the commercial value of railway operating property in the United States for the year 1904 at $11,244,852,000, while the report of the Interstate Commerce Commission shows the amount of railroad stocks and bonds not owned by railway corporations in 1904 to have been $9,585,467,711.

As to Corporations in General.

The press and the general public seem to think that some corporations are good and others bad, and to very generally class the small ones as good and the large ones as bad.

All corporations, Federal and State, are technically artificial persons and citizens created by law. So long as they conform thereto, they have the undoubted right to continue to exist and to perform all their lawful functions. They are a necessary means to our civilization, were known to our laws in Colonial times and have been ever since. They have no souls, nor are they moral beings, Strictly speaking, they can be neither bad nor good. In short. they are tools, and as such may be used well or ill.

To illustrate: If a farmer uses his spade to till his land, his act is good and should be commended; if, however, using the same spade as a weapon, he assaults his neighbor therewith, his act is evil and should be condemned. In neither case is the spade to be praised or blamed.

As a people, we have no quarrel with, nor cause of complaint against corporations as such, however clearly we may see that some, and indeed not a few of them, both big and little, have been and are maladministered in contravention alike of statute law and of common honesty. Our duty as a people is, regardless of persons, to address ourselves to detecting and punishing all violations of the Federal and of the State laws governing corporations, to the end that we shall not become, either at home or abroad, justly subject to the criticism of allowing evils of which we are so patently aware, to longer go uncorrected.

AUTOXIDATION OF ORGANIC COMPOUNDS.

By K. GEORGE FALK.

Autoxidation may be defined as a chemical reaction which takes place between a substance and free oxygen with appreciable velocity at the ordinary temperature, no catalyst being present. The matter will not be taken up historically here, partly because a number of theories have been advanced by the different workers in this field (Schönbein, Traube, Bach, Engler and others), all these theories being doubtless true to a greater or less extent, but tending to confuse the subject; and partly because the matter has been presented historically by Engler and Weissberg * and by Mellor † in such a form that this manner of treatment would simply amount to a repetition in an abbreviated form of their accounts. It has therefore been thought better to take up the question from the experimental side. The general classification of autoxidation given by Engler and Weissberg will, however, be adhered to, but only the conclusions which appear to be justified directly by the experiments will be discussed.

From the definition given above, it is evident that autoxidation may be considered as a special case of combustion — that is, of the union of a substance with oxygen which is obtained from a compound or which exists free. Combustions, as in fact all reactions, may be divided into three classes: ‡

1. Those reactions which take place of themselves.

2. Those which take place only when some exterior force overcomes the resistance which prevents the reaction from taking place.

3. Those which proceed in one direction under certain conditions and in the opposite direction under different conditions.

Strictly speaking, all reactions and therefore all oxidations, belong to the third class which regards them simply as equilibria and

* "Kritische Studien über die Vorgänge der Autoxydation." This book has been freely used in the preparation of this paper.

† "Chemical Statics and Dynamics," pp. 304–339.

‡ Cf. Bodländer, "Ueber langsame Verbrennung," "Ahrens' Sammlung chemischer und chemisch-technischer Vorträge," 3, 385 (1899).

according to which, reactions which take place under one set of conditions must necessarily take place under every set of conditions, the extent to which the reaction takes place being the only factor to change. Classes 1 and 2 are convenient subdivisions, for with given conditions, most reactions proceed practically entirely in one direction, the reason for this apparent contradiction being the difficulty of measuring the minute quantities of substances which do not react or which react in the opposite sense. If this distinction is kept clearly in mind, it will be seen that autoxidations are not divided sharply from other oxidations which take place at high temperatures, and that it would be difficult to decide in some cases whether a reaction is to be classed as an autoxidation or not. The reaction velocity is an important factor and helps to throw light upon the question whether a compound is to be considered as being oxidized by free oxygen or whether the reaction is so slow that it must be put in the second class, as not taking place under the ordinary conditions.

In the following pages, the reactions in which organic compounds figure will be chiefly taken up, as it is with these that it has been possible to study the course of many reactions by the isolation of so-called "intermediate products" which are in reality often the primary products of the reaction. With inorganic compounds, it has generally been only possible to obtain the final products, "intermediate products," if formed, tending to react or to change immediately to the most stable forms.

Not every organic compound which has been found to react with free oxygen will be taken up, but only those in which it has been possible to isolate some definite products which appear to throw light upon the course of the reaction.

Oxygen may act in three forms, O, O_2, and O_3. The atomic state, O, does not enter into the question of autoxidation, as it is only formed under special conditions, such as at high temperatures, by electrolysis, probably to a certain extent by the action of light, etc. The molecular form O_2 is the one with which autoxidations have to do chiefly, although during the last few years, Harries and his students have made a study of the action of ozone, O_3, on many organic compounds, and have arrived at some very interesting results.

To take up the question of molecular oxygen (O_2) first; the autoxidations in which it takes part may be divided into two

classes, direct and indirect. In direct autoxidations, the oxygen molecule is added to a molecule of the substance which is being oxidized, a peroxide being formed. The changes which this peroxide may undergo are determined by the conditions of the experiment and the character of the peroxide. In general it may be stated that the peroxides formed tend to lose part of their oxygen very easily and are therefore rather strong oxidizers so that a test for autoxidation formerly consisted in determining whether active oxygen (oxygen more active than that of the atmosphere) was present. For every molecule of oxygen added to a compound, one atom was given off in an active form, that is, just as much active oxygen is produced as is used up in oxidizing the substance under investigation. It will be seen presently that this is not a certain test for autoxidation, that in fact peroxides may be formed which are perfectly stable and that there are others of intermediate stability, and finally those which give up half of their oxygen very readily are obtained.

The simplest oxidation conceivable is the reaction between hydrogen and oxygen. All the evidence at the present time with regard to the reaction between hydrogen and oxygen points to the fact that at not too elevated temperatures (at which only surmise would be possible) hydrogen and oxygen unite to form hydrogen peroxide,* which then generally decomposes into water and oxygen. If this reaction can be carried out at the ordinary temperature in the absence of a catalyst, it would come into the category of autoxidations. In electrolysing a solution in which hydrogen is evolved, if a jet of oxygen (that is, oxygen in the molecular form) be directed against the electrode on which the hydrogen is separated it is found that this hydrogen is quantitatively converted into hydrogen peroxide.† In other words, it unites with the molecular oxygen, being itself in the atomic (nascent?) state. This is then a simple case of direct autoxidation, but this reaction is also important for the indirect autoxidations, as will be seen later.

In proceeding now to the organic compounds it is of course among the unsaturated substances that autoxidations would be most likely to be found. The most interesting unsaturated com-

*Cf. K. G. Falk, "The Ignition Temperatures of Hydrogen-Oxygen Mixtures," *J. Am. Chem. Soc.*, 28, 1517.

†W. Traube, *Ber.*, 22, 1496.

pound at the present time is triphenylmethyl. This substance takes up oxygen from the air, even when all moisture is rigidly excluded, to form triphenylmethyl peroxide $(C_6H_5)_3\,C-O-O-C\,(C_6H_5)_3$.* The composition of this peroxide is perfectly definite, even if the structure of the original triphenylmethyl has not as yet been satisfactorily elucidated, and it shows that molecular oxygen is taken up directly by the compound, no other substance, such as water, entering into the reaction. This peroxide has no oxidizing power at all. Its structure is proven by the formation of triphenylmethyl carbinol $(C_6H_5)_3\,C\cdot OH$ on reduction.

Oil of turpentine† absorbs oxygen from the air, moisture not having an effect on this reaction either, but the peroxide formed in this way loses half of its oxygen very easily and has therefore strong oxidizing powers. This is shown if substances which are easily oxidized (KI, indigo, etc.) are present, if no such substance is present the peroxide rearranges to a more stable oxide. These two examples may serve as extreme types of direct autoxidations; in the first there is no active oxygen formed, in the second active oxygen is formed, but in both, the first action is that of addition of molecular oxygen.

A number of other examples may now be rapidly passed over. Dimethyl, methyl ethyl, and methyl phenyl fulvenes‡ take up oxygen when their benzene solutions are allowed to remain in contact with the air, the first named to over 90 per cent. of the theoretical quantity if a molecule of oxygen were added to each double bond. The compounds formed are extremely insoluble and perhaps for this reason show no oxidizing power themselves. Amylene, hexylene, and trimethylethylene take up oxygen from the air,§ but on standing in contact with water, hydrogen peroxide is formed. With hexylene just as much oxygen is made active as is taken up. The further action of oxygen on these substances complicates these examples, and the oxidation products formed in this way tend to obscure the course of the reaction.

Kohler's work‖ on the autoxidation of unsaturated alcohols shows clearly the course of the reaction for these substances. He found that alcohols having the general formula $R'R''CH\cdot CR'''$:

* Gomberg, *Ber.*, 33, 3150.

† Engler and Weissberg, *Ber.*, 31, 3050.

‡ Engler and Frankenstein, *Ber.* 34, 2937.

§ Engler, *Ber.* 33, 1094.

‖ *Am. Chem. Jour.* 36, 177 and 529.

$C(OH)R^{iv}$, R′, R″, R‴, R^{iv} representing organic radicals, combine with molecular oxygen to form crystalline peroxides, which decompose when warmed in solution or heated in the dry state to form R′ R″ CH · CO · R‴, and R^{iv} CO_2H, showing that the original peroxides must be represented by the general formula

$$\begin{array}{c} O \quad - \quad O \\ \\ R'R''CH \cdot C \quad R'''.C(OH)R^{iv}. \end{array}$$

These peroxides with one exception do not lose an atom of oxygen readily, being more stable then the peroxides amylene and more like triphenylmethylperoxide in behavior. Triphenylmethylpropenolperoxide

$$\begin{array}{c} O\text{———}O \\ | \qquad\quad | \\ (C_6H_5)_2CH \cdot C(CH_3) \cdot C(OH)C_6H_5 \end{array}$$

occupies an intermediate position losing one atom of oxyen when treated with alcoholic potash.

An interesting case which has been cleared up finally by Baeyer and Villiger* is that of benzaldehyde. They showed that each molecule of benzaldehyde combined with one molecule of oxygen to form benzoyl hydrogen peroxide, $C_6H_5CO_2H$ (which was also obtained in a different way and whose composition is therefore well established). Benzoyl hydrogen peroxide is a strong oxidizing agent, every molecule losing an atom of oxygen, so that if no other substance is present, it may convert the benzaldehyde which is present during the oxidation into benzoic acid to which it is itself reduced at the same time.

It may appear as though only these reactions are being taken up in which molecular oxygen is added, but the facts show that these are the "intermediate" products formed in the reactions which have been studied most carefully. In most cases of oxidadation it has only been possible to obtain products which differ greatly from the initial substances and which result doutless from a very complicated series of reactions, but these will not be discussed here.

The oxidation of acetaldehyde as vapor was studied in van't Hoff's laboratory by Ewan,† who found that the velocity of the

* *Ber.*, 33, 1569.

† *Ztschr. f. phys. Chem.*, 16, 321.

reaction was proportioned to the pressure of the aldehyde and to the square root of the pressure of the oxygen. From this it was concluded that the oxygen molecule broke down into atoms which then reacted. The autoxidation of thiophenol * and of β-substituted hydroxylamines, † although interesting, do not help to clear up materially the course of direct autoxidation. Triethylphosphine ‡ behaves like benzaldehyde, taking up molecular oxygen and oxidizing some more phosphine by means of its own peroxide, or if a second oxidizable substance is present, giving up its active oxygen. Under these conditions, just as much oxygen is taken up by the second substance as by the triethylphosphine.

With regard to the inorganic direct autoxidations, the most evident examples are the alkali metals (Na, K, Rb and Cs) which take up oxygen directly to form peroxides. Phosphorus does not react with pure oxygen at atmospheric pressure but does under diminished pressure, ozone and hydrogen peroxide being formed. The reasons for this behavior are not definite as yet.

This list does not pretend by any means to be a complete one of the direct oxidations at ordinary temperatures which are recorded in the literature, but it may serve to show the trend the study of these phenomena is taking.

Taking up the indirect autoxidations, it has been shown that by the action of atomic hydrogen on molecular oxygen, hydrogen peroxide is formed. If a substance is introduced into water or an aqueous solution which itself gives off hydrogen or which causes hydrogen to separate from the solution to even a very minute extent, and if molecular oxygen is passed into the solution, hydrogen peroxide will be formed. This is shown clearly when metals are brought in contact with aqueous solutions. If zinc is put into dilute acid, it sends positive ions into solution, hydrogen ions from the acid giving up their charges and separating on the zinc, until a definite, though very small, quantity of zinc has gone into solution. Passing air into the solution, the oxygen combines with the hydrogen which has appeared on the zinc, but the hydrogen peroxide which is formed is decomposed by the metal into water and oxygen. As soon as this hydrogen is removed, more zinc dis-

* Engler and Frankenstein, *Ber.*, 37, 3274.

† Bamberger, *Ber.*, 33, 121, and 36, 836.

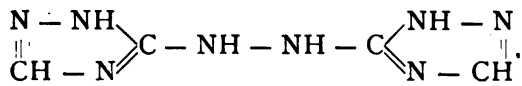
‡ Engler and Wild, *Ber.*, 30, 1672; Engler and Weissberg, *Ber.*, 31, 3056; Jorissen, *Ztschr. f. phys. Chem.*, 22, 38, and *Ber.*, 29, 1709.

solves, more hydrogen separates, and the process continues as long as oxygen is passed into the solution. If zinc amalgam is used instead of zinc, the hydrogen peroxide formed is not decomposed and remains in solution. The hydrogen which separates on the zinc is in exactly the same state as the hydrogen which separates on the cathode in electrolysis and reacts in the same way. Similar reasoning holds for the solution of gold in potassium cyanide (MacArthur extraction process), which takes place when air is present. The equations for this reaction are:

$$2Au + 4KCN = 2KAu(CN)_2 + 2K$$
$$2K + 2\dot{H} + 2\overline{OH} + O_2 = 2\dot{K} + 2\overline{OH} + H_2O_2.$$

the hydrogen and hydroxyl ions appearing as a result of the hydrolytic dissociation of the potassium cyanide. The rusting of iron, which takes place when air, carbon dioxide, and moisture are present, hydrogen peroxide being formed as one of the products, is also a case of indirect autoxidation. A number of other examples among the metals could be cited, but these will suffice. Among the organic compounds, the most prominent example is the oxidation of indigo white in aqueous solution by means of air to form indigo. In the presence of alkali, any hydrogen peroxide which may be formed is immediately used up to oxidize some more of the indigo white, but if lead dioxide is present, the hydrogen peroxide formed reduces this instead, and the total amount which would have been formed can be determined in this way. It was found * that one molecule of indigo white reacts with one molecule of oxygen to form one molecule of indigo and one of hydrogen peroxide.

Hydrazobenzene * reacts slowly with air in aqueous solution, but if pure oxygen is used the action is rapid, and quantitative measurements have shown that for each molecule of oxygen which reacts, one molecule of hydrogen peroxide is formed. The same results have been obtained with hydrazotriazole *



a number of hydrazones,* oxanthrol,† dihydrophenanthrene-qui-

* Manchot and Herzog, *Liebig's Annalen*, 316, 318.

† Biltz and others, *Liebig's Annalen*, 305, 165; 308, 1; 321, 1; 324, 310.

none,* hydrochrysoquinone * and hydroretenequinone.* With the phenols † the results are not definite as yet, as the products formed are extremely unstable and are therefore generally oxidized further. It has, however, been possible to obtain hydrogen peroxide in isolated cases (2-amido-1-naphthol † for instance), but definite conclusions cannot be arrived at as the reactions are too complex.‡

If, in considering the reaction between the hydrogen which is separated by electrolysis and molecular oxygen, the reaction is assumed to take place between the *ionic* hydrogen and the oxygen, the same must be held to be true for all the indirect autoxidations as they are exactly analogous. As it has not been possible to decide whether it is the ionic or ordinary hydrogen which reacts, the latter view has been adopted as it shows the relation between the indirect autoxidations as clearly as the other. In considering indirect autoxidations, it has been the custom to consider the water as entering into the reaction, as for instance with zinc

$$Zn + \begin{matrix} H-O \\ H-O \end{matrix} \begin{matrix} -H \\ -H \end{matrix} + O_2 = Zn(OH)_2 + H_2O_2,$$

since these reactions only take place when water (or some solvent) is present, but in the present discussion, water is looked upon as simply acting as a solvent. Although it is admitted that water may take part in the reactions, it seems better to leave it out entirely, because just as satisfactory and perhaps a less confusing classification can be obtained in this way.

To sum up, direct autoxidation consists in the addition of molecular oxygen to the substance under investigation, indirect in the addition of molecular oxygen to the dissociated hydrogen of the substance under investigation which thus loses part of its hydrogen. If both reactions are possible with the same substance, the one which proceeds with the greater velocity under the given conditions will be the one to take place.

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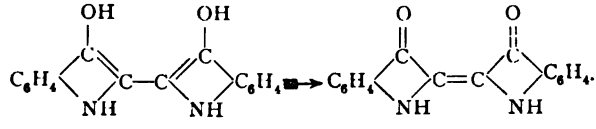
* Manchot, *Liebig's Annalen* 314, 179.

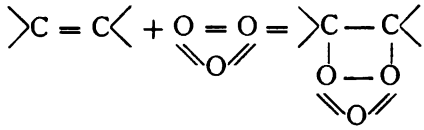
† Berthelot, *Comptes Rendus*, 126, 1066 and 1459. Harries, *Ber.* 35, 2954.

‡ Indigo white may be considered as a phenol which on oxidation yields a quinone



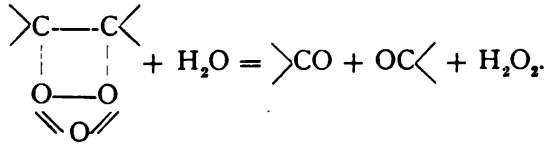
Although ozone does not occur under ordinary conditions in the atmosphere or with free oxygen, some of its reactions may be mentioned briefly here as they are similar to those of oxygen. Harries * was able to obtain products by the action of ozone on unsaturated organic compounds which were formed by the direct union of the ozone and the organic substance. He studied the unsaturated aliphatic and aromatic hydrocarbons, alcohols, aldehydes, ketones, and acids. His results may be summarized asfollows:

A molecule of ozone is added to each double bond present in the compound, the products formed being called ozonides.



(Ozone is formulated O = O by Harries.)

These ozonides are thick syrupy liquids, very explosive, and strong oxidizing agents. They are monomolecular, as shown by the freezing point lowering in glacial acetic acid and in benzene. With water the ozonides decompose with more or less rapidity according to the equation:



Ketones, aldehydes and acids, besides adding a molecule of ozone to each double bond, also add another atom of oxygen to the keto group, the ozone breaking down into three atoms of oxygen under these conditions.

The special cases which Harries has worked upon, the preparation of new compounds, and the work upon the structure of many unsaturated compounds, chiefly cyclic, which resulted from the use of ozone, although of the greatest interest, do not fall within the scope of this paper. The general results have, therefore, only been given; for the details of the work, the original papers must be consulted.

HAVEMEYER LABORATORIES,
COLUMBIA UNIVERSITY,
April, 1907.

* Liebig's *Annalen*, 343, 311.

THE GNOMONIC PROJECTION FROM A GRAPHICAL STANDPOINT.

By AUSTIN F. ROGERS.

Since the production of the series of protractors designed by the late Professor Penfield* perhaps the stereographic projection affords, all things considered, the best representation of the faces, angles and zones of crystals and the best basis for the application of graphical methods. Yet the gnomonic projection has certain advantages over the stereographic† and it may often be employed in preference to the stereographic. As usually developed,‡ the gnomonic projection is intimately connected with the polar symbols and polar elements of Goldschmidt, but it may also be used with the Miller indices and usually accepted axial elements, a, b, c, and α, β, γ, as will be the purpose of this article to show. Some of the graphic solutions are new, but opportunity is taken at the same time to include what are thought to be the best methods for the solution of most of the problems of determinative crystallography. It may be convenient to have these various methods collected in one place and described in English.

Principle of the Gnomonic Projection.

The plane of projection is a plane perpendicular to the c-axis of the crystal and at a unit's distance from the center. This plane is parallel to the *001* face in all except the monoclinic and triclinic systems. From the center of the crystal lines are drawn normal to the crystal faces and extended to cut the plane of projection. The

* *Amer. Jour. Sci.* (4), 11, 115, 1901.

† The comparative advantages of the stereographic and gnomonic projections may be summarized thus:

Stereographic.	Gnomonic.
1. Small sheets suffice.	1. Requires large sheets.
2. Zones are arcs of circles.	2. Zones are straight lines.
3. Interfacial angles easily obtained.	3. Interfacial angles difficult to obtain.
4. Indices not obtained directly.	4. Indices obtained directly.
5. Projection includes all forms in upper half of crystal.	5. *100 010 hk0's* and other steep forms do not appear in the projection.

‡ Goldschmidt, *Zeit. f. Kryst. u. Min.*, 21, 210, 1893.
Eakle, *Bull. Dept. Geol. Univ. of Cal.*, 3, 31, 1902.

points thus determined constitute the gnomonic projection. The distances of the points from the center of the projection are equal to the tangents of the corresponding angles. Therefore faces in the zone [*100* : *hko* : *010*] are projected at infinity and do not appear in the projection proper. They are usually indicated at the edge of the drawing by the direction of their normals, as in Fig. 1.

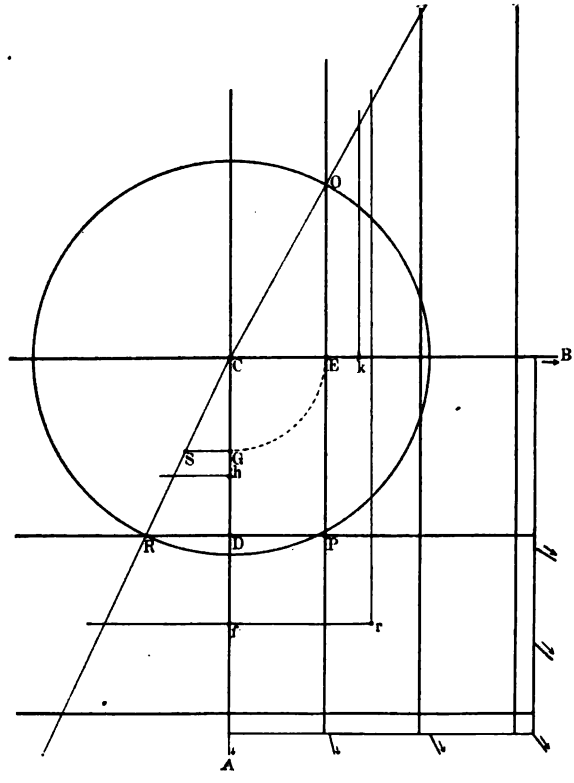


FIG. 1. Gnomonic projection of an orthorhombic crystal, topaz. $c = CE$ in terms of radius; $a = CS$ in terms of radius. The $h : l$ scale line is CR; the $k : l$ scale line is CO. $C(001)$, $D(101)$, $E(011)$, $P(111)$, $A(100)$, $B(010)$.

For the same reason very steep pyramids, domes, etc., do not appear in the projection. In the gnomonic projection zones are represented by straight lines connecting the points of the projection and in this fact lies one of the principal advantages of the projection.

The principal classes of zones are (1) zones through *001* which are radial for all systems except monoclinic and triclinic; for faces

in any one of these zones the ratio $h:k$ is a constant; (2) zones through *100* which are all parallel to the zone [*001*:*hol*: *100*]; for faces of these zones the ratio $k:l$ is a constant; (3) zones through *010* which are all parallel to the zone [*001*: *okl*: *010*]; for faces of these zones the ratio $h:l$ is a constant.

If the principal zone lines through *100* and *010* (*i. e.*, those through *011*, *021*, *031*, etc., and *101*, *201*, *301*, etc.) are drawn we obtain a network of parallelograms. These vary in shape for the different crystal systems. For the isometric and tetragonal systems the network is composed of squares (the isometric invariably one of 45 gnomonic degrees, the tetragonal varying for different substances), for orthorhombic and monoclinic systems, of rectangles, for the hexagonal (and trigonal), of rhombuses and for the triclinic, of rhomboids.

Plotting of the Gnomonic Projection.

The usual method of plotting the rectangular coördinates on millimeter cross-section paper may be simplified by laying off the coördinate angles φ and ρ * directly on the sheets devised by Penfield † for the stereographic projection. φ, the angle between the prime meridian (line from center to *010*) and a radius for face in question is laid off on the divided circle. ρ, the angle from the center along this radius, is given by the tangent scale No. 2 at the top of the sheet. It is convenient to cut this scale out and use it as a protractor. The divided circle is of unit radius and is the locus of all faces 45° from center of projection. On these sheets faces with ρ greater than 68 to 76° can not be included in the drawing. Faces in the zone of *hko's* can be indicated either on divided circle or on edge of sheet as in Fig. 1.

Graphic Determination of the Indices.

The principal advantage of the gnomonic projection is that the indices may be determined directly from the projection. For example, the distance of an *hol* face from *001* is equal to h/l times the distance of *101* from *001* and this is true along whatever line the distance may be measured. One method is simply to measure off the distances with a millimeter scale along the zone lines. This

* The gnomonic projection is particularly adapted for use with two-circle measurements. Plotting from one-circle measurements is more difficult. It is facilitated by the use of the gnomonic net (see later).

† *Loc. cit.*

introduces numerical calculations with a chance of error. So a line *CR* is drawn (fig. 1) through *C*(*001*) cutting the *hol* zone lines so that the distance from *C*(*001*) to the zone line through *D* (*101*) is equal to the radius of the circle. Then the ratios *h* : *l* of both *hol* and *hkl* face may be measured and read off on this line by means of scale no. 4 of the sheets. Similarly a line *CO* is drawn, cutting the *okl* zone lines so that *CO* is equal to the radius of the circle. Then the ratios *k* : *l* of *okl* and *hkl* faces may be read off.

The two ratios *h* : *l* and *k* : *l* determine *hkl*. Fig. 1 shows these two "scale lines" for an orthorhombic crystal topaz. *CR* is the scale line for *h* : *l* ratio and *CO* for *k* : *l* ratio. In the figure *k* is *043*; *h*, *203*; *f*, *302*: and *r*, *332*. The *hko* faces indicated by arrows

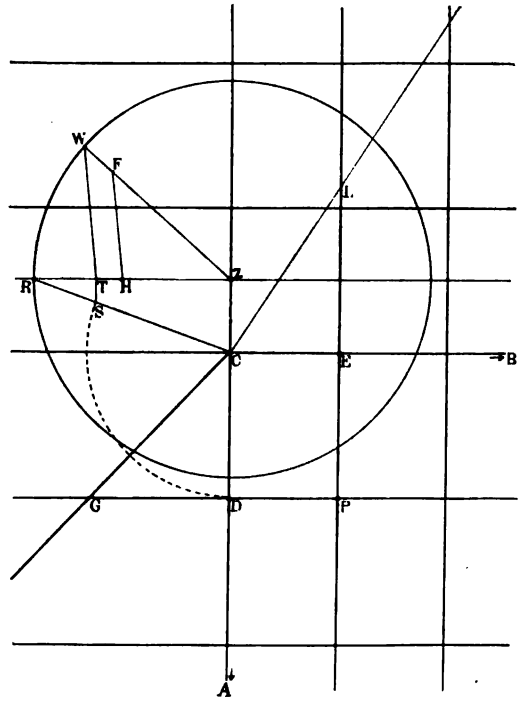


FIG. 2. Gnomonic projection of a monoclinic crystal, colemanite. $c = CE$ in terms of radius; $a = FZ$ in terms of radius; $90° - \beta = CZ$ (gnomonic degrees). The *h* : *l* scale line is *CL*; the *k* : *l* scale line *CG*. *C*(001), *A*(100), *B*(010), *D*(101), *E*(011), *P*(111).

are in order from *A* (*100*) to *B* (*010*):—*210*, *110*, *230*, *120*, *130*. The *hko* faces may be determined from the ratios *h* : *k* of corresponding *hkl*'s.

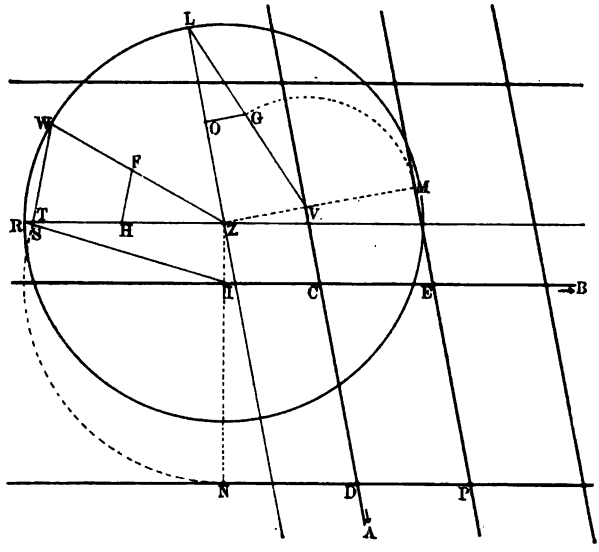


FIG. 3. Gnomonic projection of a triclinic crystal, chalcanthite; $c = ZO$ in terms of radius: $a = ZF$ in terms of radius; $\alpha = 90° + ZV$ (gnomonic degrees); $\beta = 90° + ZI$ (gnomonic degrees). The scale lines are not drawn. $A(100)$, $B(010)$, $C(001)$, $D(101)$, $E(011)$, $P(111)$.

GRAPHIC DETERMINATION OF THE AXIAL ELEMENTS.

In the *tetragonal* system c is the distance of 011 from center of projection 001 in terms of the radius. (Use scale no. 4.)

In the *hexagonal* system c is the distance of $11\bar{2}1$ from the center of projection 0001.

In the *orthorhombic* system c is the distance of $E(011)$ from $C(001)$. For determination of a, reference is made to Fig. 1. The distance of $D(101)$ from $C(001)$ is c/a. On CD lay off CG equal to CE. Draw GS parallel to DR, R being the intersection of h/e scale line with zone line $[D(101) : 111 : 010]$. Then CS is equal to a in terms of radius CR.

In the *monoclinic* system (Fig. 2) c is the distance of $E(011)$ from $C(001)$. The distance of $D(101)$ from $C(001)$ in terms of CR is equal to c/a. On CR lay off CS equal to CD. Draw $S1$ parallel to CZ. On ZR lay off ZH equal to CE. On any radius ZW, draw HF parallel to TW. Then ZF is equal to a in terms of radius. $ZC = 90 - \beta$ in gnomonic degrees.

In the *triclinic* system the determination of the elements is more complicated as indeed are all the relations of the triclinic system. In Fig. 3, the faces are $C(001)$; $E(011)$; $D(101)$; $P(111)$; $A(100)$

and B (010). Draw ZL, ZR and ZM, ZN lines respectively parallel and perpendicular to the prominent zone lines. On VL lay off VG equal to VM. Draw GO parallel to VZ. Then ZO is equal to c in terms of the radius. On IR lay off IS equal to IN. Draw ST parallel to IZ. Then $ZT = c/a$. Lay off ZH equal to ZO (c). On any radius ZW draw FH parallel to TW. Then $ZF = a$ in terms of radius. $90 \pm ZV$ is equal to α and $90° \pm ZI$ is equal to β. To determine γ, the plane of the projection is changed making C(001) the center and the angle between the two radial zones [001 : 010] and [001 : 100] read off. A method for doing this is described later. For example in Fig. 6, if P is 001, the angle $R'P'S'$ obtained after the transformation is equal to γ.

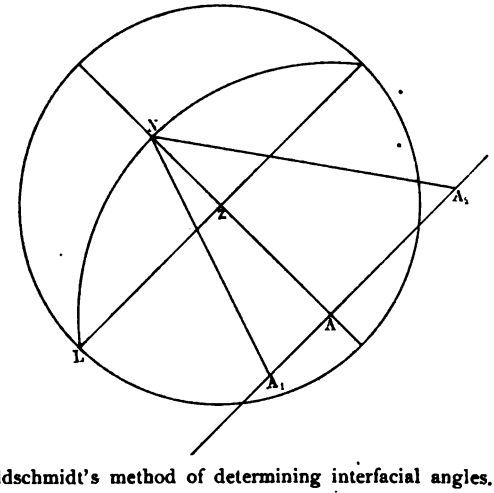


FIG. 4. Goldschmidt's method of determining interfacial angles. The angle between A_1 and A_2 is the angle A_1NA_2, N being the angle point of the zone $[A_1A_2]$.

INTERFACIAL ANGLES ON THE GNOMONIC PROJECTION.

A method of determining interfacial angles on a gnomonic projection has been described by Goldschmidt.* In Fig. 4 to determine the angle between the faces A_1 and A_2 draw the zone line through them. Draw diameter perpendicular to A_1A_2 at A. Find N, intersection of diameter through A and an arc of a circle with radius AL and center, A. Then A_1NA_2 is the required angle. Goldschmidt calls N the angle point of the zone. All faces on the zone may be determined at this point. It may be found from the fact that ZN (in stereographic degrees) is the complement of ZA (in gnomonic degrees).

* "Ueber Projection und graphische Krystallberechnung," Berlin, 1877.

Interfacial angles may also be determined by means of what Hilton * calls the gnomonic net, which is simply a combination of the gnomonic projections of great circles and small vertical circles corresponding to the parallels of latitude and longitude on the earth's surface.

Fig. 5 represents a gnomonic net of one quadrant for every five degrees. This was constructed by getting a number of points by

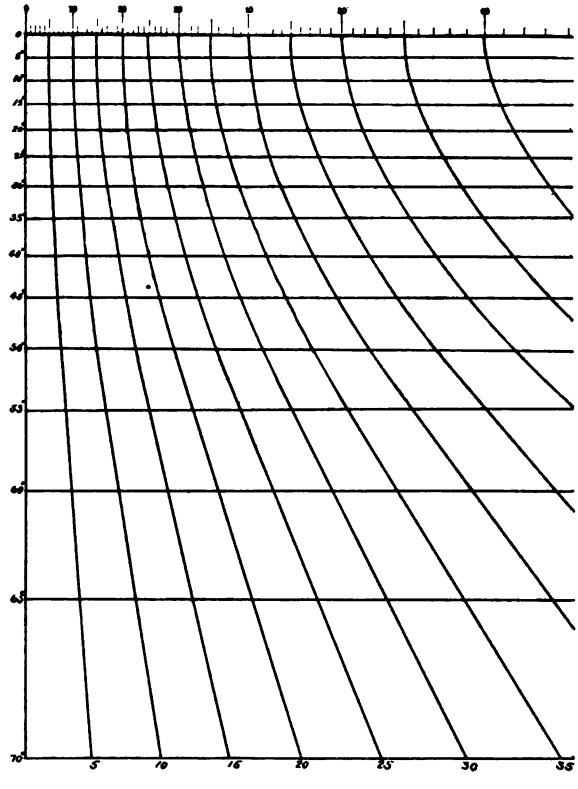


FIG. 5. The gnomonic net of Hilton. One quadrant of the gnomonic projection of small vertical circles and great circles for every five degrees.

the method of Goldschmidt just described and connecting them with curves. The upper left-hand corner of the net is to be placed at the center of the projection. Then if a given zone line is placed on or parallel to the straight lines (projection of great circles) in-

* *Min. Mag.*, 14, 18, 1904.

terfacial angles of faces on that zone are read off on the parabolic lines (projections of small vertical circles).

Zone Angles on the Gnomonic Projection.

The angle between two zones is one of three kinds, (1) Radial zones intersecting at the center which meet at their true angle. (2) The angle between parallel zones is the perpendicular distance between the zone lines in gnomonic degrees. (3) In general the angle between two zones may be found, as suggested by Hilton,* by changing the plane of projection so that their intersection becomes the center of the projection. The simplest way is to take

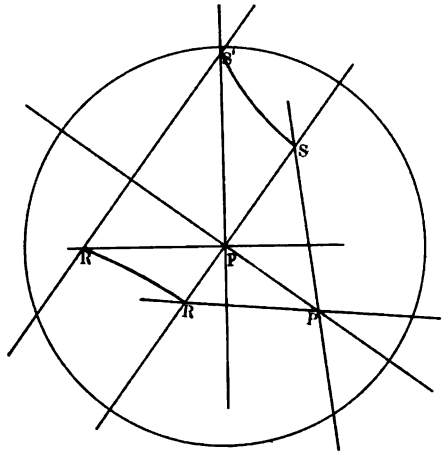


Fig. 6. Method of determining zone angle by changing the plane of projection so that P, the intersection of the two zones, becomes the center P'. The edge RPS becomes $R'P'S'$, the true zone angle.

the intersections of the zones with the line normal to the line joining the old and new centers of the projection. Thus in Fig. 6 to determine the angle between zones PR and PS shift the points P, R, and S so that P becomes the new center P^1. P moves the distance PP' in gnomonic degrees, along a straight line. R and S move the same number of degrees but along parabolas. The new points R' and S' are easily found by using the gnomonic net. Then the angle $R'P'S'$ is the angle between the two zones.

* *Loc. cit.*

Crystal Drawing.

To Goldschmidt* we owe a simple method of obtaining a clinographic projection from gnomonic projection. The plane of projection is usually a plane obtained by revolving the vertical plane perpendicularly to 010 18° 30′ to the right and tipping back 9° 30′. The trace of the plane of projection is the dot-and-dash line in Fig.

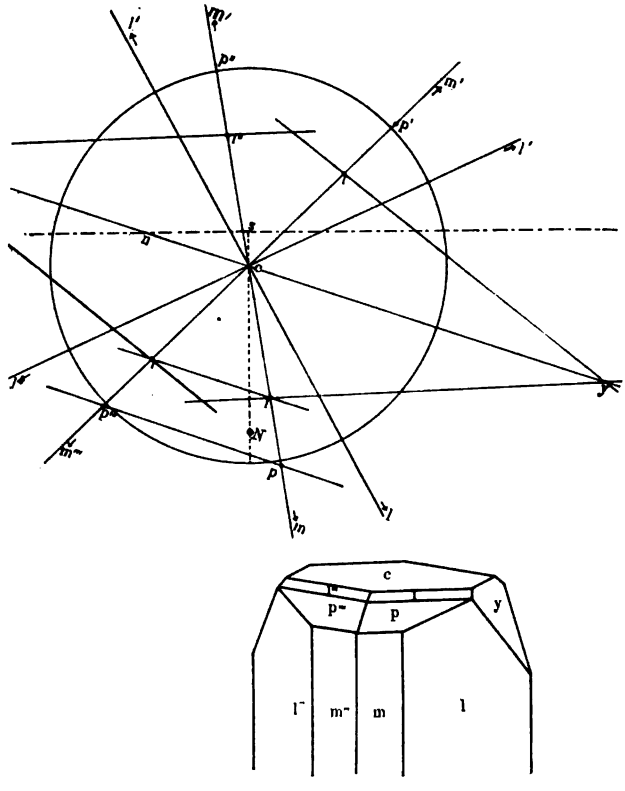


Fig. 7. Goldschmidt's method of making a crystal drawing (topaz) from a gnomonic projection. — The dot-and-dash line is the trace of the plane of projection. N is the angle point of this trace. $c(001)$, $m(110)$, $l(120)$, $y(041)$, $p(111)$, $i(223)$.

7, where the angle *snc* is 18° 30′ and the angle *cs* is 9° 30′ (gnomonic degrees). The angle point of this trace is N, which is 80° 30′ (stereographic degrees) from the center. The desired edge direction of two faces is a line normal to the line joining N and the intersection of the zone line of the two faces with the trace of

* *Zeit. f. Kryst. u. Min.*, 19 : 352, 1891.

the plane of projection. For example, the edge direction $c:y$ is a line normal to the line nN (not drawn). The edge direction of the faces in the zone [*100* : *hko* : *010*] is a line normal to the trace, *sn*. Fig. *7* represents a clinographic projection of a topaz crystal (*c*, *001*; *m*, *110*; *l*, *120*; *y*, *041*; *i*, *223*; *p*, *111*) made from the accompanying gnomonic projection. Such a projection is identical with one made from a stereographic projection.

A combination of an orthographic projection and stereographic projection may be drawn from the gnomonic projection as described by Penfield. *

STANFORD UNIVERSITY, CAL., Jan., 1907.

* *Amer. Jour. Sci.* (4), 21; 206, 1906.

MINE SURVEYING.

With Special Reference to Shaft Surveying.

By C. E. MORRISON.

Mine surveying, and by this is meant the surveying required of the mine surveyor, includes in its most general form not only all of those methods which apply to surface operations, but also such modifications of the same as must be adopted with the changed conditions underground to secure similar data.

It therefore comprises land, topographic, hydrographic and even on occasion geodetic work, as may readily be seen in the case where the properties held by the mining company are so extensive as to necessitate detailed surveys of large tracts of land. It frequently happens that the holdings include a number of mining claims, placer locations, mill sites, etc., and where accurate records are desired, one or all of the above methods of surveying may be demanded.

We might then divide mine surveying into its two most natural parts, surface and subsurface operations, considering perhaps, as an intermediate step, the work of tying the two together.

The surface surveys are two in number: first, and the more important, a land survey together with its map showing by a convenient scale all property, claim and building lines, the position of monuments, shafts, surface workings, roads, etc.; and second, a topographic survey and map representing the conformation of the land, all natural and artificial features such as hills and valleys, streams, roads, railroads, buildings, mine hoists, surface workings, etc.; in fact all detail usually required in such work.

Reference monuments should be established near the shaft or tunnel to which the surface survey is tied in, so that a line may be easily carried into the mine, and the subsurface surveys thus joined to those on top. These monuments best take the form of rectangular stone blocks about four feet in length and six by six inches in section. They should be sunk into the ground below frost line to preclude movement due to successive freezing and thawing, and should have their top surfaces carefully dressed, with a hole in the

center to receive a copper plug on which the reference point is to be marked. There should be at least three such monuments, arranged in the form of a triangle and situated somewhat closely to the shaft down which the survey is to be carried. The triangle arrangement is much better than any other since the slightest movement of the monuments will be apparent not only in the angles, but in the distances: and the locality of the shaft itself is good since the ground is less likely to cave there than at any other place. In fact the position of the shaft is always chosen with the view of having it in solid ground and with few workings below, to avoid the possibility of its being affected.

An excellent arrangement of the points with regard to the shaft is shown in Fig. 1, for with *B* and *C* selected as they are, it

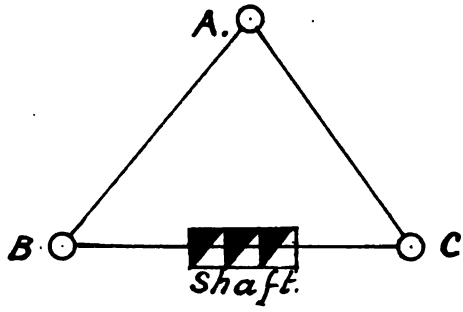


FIG. 1.

will be easy to get a line into the mine. *A*, *B* and *C* are not only tied into the rest of the surface work, but all the included angles, and the sides of the triangle are measured, each station being occupied successively and the angles read by repetition. The azimuth of at least one of the sides is determined either by a sight on Polaris, the Solar attachment, or a direct observation on the sun.

It is claimed that underground surveys are never so accurate as those made above ground, but this is not necessarily true. It is a fact however, that with the generally cramped conditions of the set up, the somewhat unstable condition of the tripod and the use of artificial light, it becomes more difficult to secure accuracy than where these do not obtain, but care and practice will usually overcome these inconveniences to such an extent as to make it possible to run an underground traverse with the same degree of precision as is required on the surface.

The artificial light is a hindrance. Candles must be used to illuminate the crosshairs of the telescope and also to illuminate the backsight and foresight. In sighting at the former the telescope is directed at the point, while the instrumentman holds his candle in front of, and to the side of, the objective glass so that enough light goes down the tube to show the wires as two dark lines, and yet not enough to obscure his sight. To make the back sight or foresight visible, a special form of lamp is used, the flame of which is vertically below the station from which the lamp hangs.* The flame is bisected for a sight. If lamps are not used then the plumb bob is hung from the station and the string made visible by holding behind it a piece of paper or tracing cloth, at the back of which is placed a candle. Such a sight may be seen under favorable conditions for distances as great as it is convenient to measure with one set up, that is from 300 to 400 feet. Where the mine is smoky, however, the plummet lamp may be seen further, but is necessarily not so accurate, since the bisection of the flame is less precise than the bisection of a plumb line.

It is usually customary to run the drifts and headings on the levels in advance of the stoping as a means of prospecting the ground and hence the location of these workings must be plotted from the survey notes to indicate intelligibly the relation of the ore bodies to each other. The surveys also form a basis from which to continue further work, and to check work done in the stopes. Generally such surveys are made at least once a month so that the maps may be plotted and kept up to date, and are therefore carried out to the face of every working on the level, *i. e.*, crosscut and drift, at these times.

The stations used in subsurface work differ somewhat both in form and the manner of location, from those on the surface. Where the mine is pretty generally timbered a good form of station is a small round brass tag, numbered, and with holes in it so that nails may be driven through to fasten it to the cap or girt: or where no timbers exist a hole must be drilled in the roof, a plug inserted and the station attached to this. The latter requires time to drill the hole, however, and is inconvenient also when the roof is high. The number makes it easy to reference in the field notes and also serves both on the map and in the mine as a means of readily locating oneself. In some cases a white paint mark is

* See *Mines and Minerals*, Vol. XXV., No 1, August, 1904.

used to distinguish the station, or a spud, that is a horseshoe nail with a hole in the head for the plumb bob string, is used and driven into a crack in the roof or into the timbers.

The selection of the points where stations should be placed needs some care, for frequently the ground will shift, or actually be removed in the course of mining operations, requiring relocation or perhaps introducing an unknown error. At the same time, it should be remembered, that the object is to get as long a sight as possible and as many sights from the one set up as is consistent with accuracy and the time required. For example, a station located at the intersection of several workings is good for a line of sight down each one and hence may save one or many set ups.

A difficulty, at least to inexperienced men, is the centering of the transit under a point rather than over one. The telescope carries a center mark on the top so that when the instrument is in adjustment and the long bubble in the center of its tube, the point on top is in the vertical axis of the instrument. The transit therefore must be centered under the point of the plumb bob which is suspended from the roof.

In measuring angles, because of the difficulties of light, cramped quarters, traffic, etc., it is always better to read the angle once from the zero around the scale, and then double to check. At the same time the magnetic bearing should be recorded to check the azimuth. It is probably much the best way to read each angle from a zero vernier rather than to run a continuous azimuth, since in the latter case errors may creep in unobserved.

The elevations in underground work are determined by measuring down from the known elevation of the back sight to the horizontal line of sight and then up from this line, which is the H.I., to any point whose elevation is desired. In surface surveys just the opposite is the procedure.

The intermediate step in a mine survey is that of connecting the reference points or lines on the surface with those in the mine. All mines are entered either by a tunnel or a shaft. In the former case no difficulty presents itself, as the entrance from the surface is almost horizontal, the only changed conditions being the darkness, and the necessity of setting up underneath a point in the roof.

Where the entrance, however, is by a shaft, whether inclined or

vertical, some trouble results, and a good deal of care is required to bring the survey from the surface into the mine.

We will first discuss the means of doing this where the shaft is vertical and then take up the case where it is inclined.

With the former condition, two cases may arise; the mine may be entered by two vertical shafts some distance from each other connected beneath the surface, or there may be but one vertical shaft to enter the workings.

With the former condition the method is as follows: At each shaft a single wire, either iron or steel, rather fine and capable of supporting a weight of from ten to twenty pounds, is unwound from its reel, which rests on planking over the center of the compartment and at such a height above the floor of the shaft-house that when the wire hangs true and vertical it may be located by a course and horizontal distance from the transit set up.

With a small weight attached to keep it approximately vertical, the wire is lowered in the shaft until it reaches the station of the level where the connection is to be made. Here the weight is detached. Across the shaft some planking is laid, and upon this is placed a bucket of water which is to receive the heavier ten to twenty-pound weight that is now attached to the wire.

This heavier weight is necessary to insure absolute verticality and is allowed to hang suspended in the water, care being taken to see that it touches neither the sides nor the bottom of the bucket, so that the tendency to swing, caused by the air currents in the shaft, may be reduced to a minimum.

When the wire is steady and everything is in readiness, the man below passes his lighted candle completely around it, and if the light is continuously visible to the man above, it will prove that the wire is hanging perfectly free and not touching the wall plates of the shaft. Satisfied that this is the case, the man on top locates from his set-up, by a course and horizontal distance, the position of the vertical wire, and thus makes it part of the surface survey. He then picks up, goes down the shaft either in the cage in another compartment or by a ladder way, and sets up his instrument at some convenient point in the station of the level where the connection is to be made. This set-up will be only temporary, and the point — probably a mark on the floor — will be maintained only for the purpose of a back sight.

Having set up here, the survey on the level now begins. With

the vernier reading zero, a back sight is taken on the wire in the shaft, the horizontal distance measured to it, and the instrument is then swung in azimuth to sight a plumb line hanging from the first permanent station set, which may be a nail in a cap, girt or wooden plug in the roof.

The angle is recorded, doubled, that is reread to check the first reading, the magnetic bearing taken as a check on the azimuth to be computed later, and the horizontal distance measured to the station.

The transit is then picked up, set under the station just located, and with a B.S. on the temporary station the same operation is

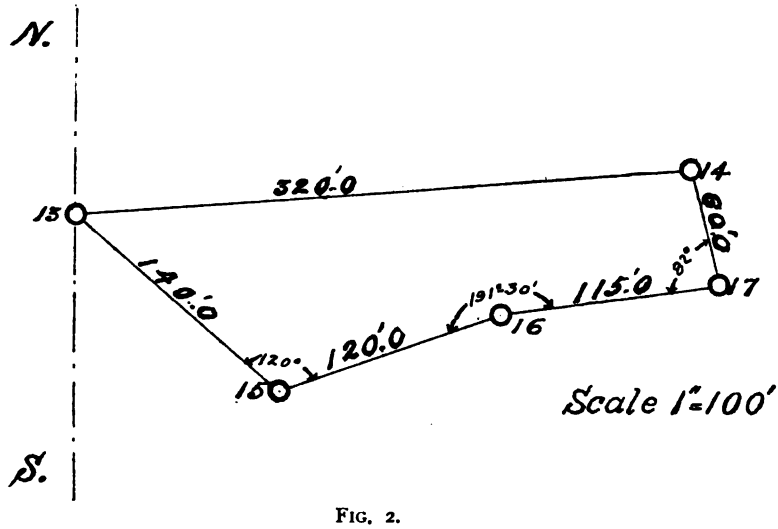


FIG. 2.

gone through with to locate the next. Similarly all the other stations are located, and finally the second wire whose position has been determined on the surface is tied in to the underground survey. The field work is now completed.

Let us assume that the following set of notes represents the work done in the entire survey, and that Fig. 2 represents to scale the location of the different points in the traverse.

Station.	Angle.	Magnetic Reading.	— Height.	+ Height.	Vertical Angle.	Floor Elevation.	Horizontal Distance.
15 B.S. 13	120°-00′	.....	398.′5	3.′4	.....	6.′5	120.′0
.....	240°-00′	.....	.....	.....	.....	.....	.....
.....	.....	.....	.....	.....	.....	.....	.....
.....	.....	.....	.....	.....	.....	.....	.....
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.....	.....	.....	.....	.....	.....	.....	.....
.....	.....	.....	.....	.....	.....	.....	.....
16 B.S. 15	191°-30′	.....	2.′6	3.′1	.....	7.′2	115.′0
.....	23°-00′	.....	.....	.....	.....	.....	.....
17 B.S. 16	82°-00′	.....	3.′2	.....	.....	.....	60.′0
.....	164°-00′	.....	.....	.....	.....	.....	.....

Remarks.

To station 16. A nail west face center 5.′0 cap.

Horizontal distance from 13 (wire in shaft) to 15 (set up) 140.′0. Vertical distance measured down shaft from H.I. of surface set up to H.I. of set up at station 15 = 398.′5.

Station 15 is a tack in a cap at the entrance of cross cut east.

To station 17 nail in 7.′0 cap, west face 2.′0 from south post.

To wire in shaft (14). Vertical distance from H.I. at station 17 up wire, to H.I. of surface set up, 401.′0.

The above notes are made on the assumption that 31 and 14 have been definitely located in the surface survey with respect to a system of rectangular coördinates whose origin is so far west and south that the axes will include all of the surveys to be made.

Let the coördinates of 13 and 14 be

	N.	E.
Sta. 13.	4372.65	7621.32
Sta. 14.	4400.35	7937.85

These values of 13 and 14 will enable us to determine the true bearing of the line, for, taking the difference in latitude and longitude, 27.70 and 316.53 or northing and easting, from the right triangle 13–x–14, we may compute the tangent of the angle x–14–13 which is the bearing of the line 14–13. This is found to be S. 85°–W.

$$\frac{316.53}{27.70} = \tan x,\ 14,\ 13$$

$$\log\ 316.53 = 2.5004149$$
$$\log\ 27.70 = 1.4424798$$
$$1.0579351 = 85°.$$

Looking at the figure it will be seen that we now have the true bearing of 13–14, the length of each course, and all the included

angles of the traverse except those of which each wire is an apex. Until we can determine these angles it will be impossible for us to get the true bearings of the other lines or courses. Let us assume, however, to solve this, that the azimuth of 13–15 is 180°, *i. e.*, the bearing south; the coördinates of 13 are known. Then from the angles turned we may calculate the bearings of all the other courses referred to 13–15, and the coördinates of all the stations. But this determines the coördinates of 14 with respect to 13 under the assumed conditions and hence gives us a new bearing for 13–14. This being so, we are able to calculate the included angles at both 13 and 14, and with these determined we may recalculate the entire survey and relocate the points under ground with reference to the true bearing of 13–14.

The computations would be as follows:

Assumed bearing of course 13–15, south; distance 140′.0.

			Coördinates. N.	E.
		(13)	4372.65	7621.32
13–15	180°–00′ azimuth.		— 140.00	0.00
	120°–00′ angle turned.	(15)	4232.65	7621.32
	300°–00′		— 60.00	+ 103.95
	180°–00′			
15–16	120°–00′ azimuth S. 60° E. 120.′0	(16)	4172.65	7725.27
	191°–30′ angle turned		— 76.20	+ 86.13
	311°–30′			
	180°–00′			
16–17	131°–30′ azimuth S. 48° 30′ E. 115.′0	(17)	4096.45	7811.40
	82°–00′ angle turned		+ 50.03	+ 33.12
	213°–30′			
	180°–00′			
		(14)	4146.48	7844.52
17–14	33°–30′ azimuth N. 33°–30 E. 60.′0			

In these calculations traverse tables may be conveniently used for the determination of the coördinates of the different points. From these results we are enabled to get the differences in latitude and departure of stations 13 and 14 on the assumption that course 13–15 runs due south.

	N.	E.
Sta. 13.	4372.65	7621.32
Sta. 14.	4146.48	7844.52
	226.17	223.20

These differences then give the bearing of course 14–13 as N. 44°–37′ W. and hence an azimuth of 315°–23′.

The angle at 13, since 15 is assumed to bear south, is therefore equal to 44°–37′ while the angle at 14, because 17 bears S. 33°–30′ W., is equal to 101°–53′.

Hence all the angles in the traverse are known and we may now use the true azimuth for 13–14 and calculate the true azimuth and bearing of each course and the true coördinates of each station. The calculations follow.

The true bearing of course 14–13 is S. 85° W. and the distance, 320′.00 feet.

				Coördinates. N.	Coördinates. E.
14–13	265°–00′	azimuth	(13)	4372.65	7621.32
	44°–37′	angle turned		— 89.27	+107.85
	319–37′				
	180–00′				
13–15	129°–37′	azimuth S. 50°–23′ E. 140.′0	(15)	4283.38	7729.17
	120°–00′	angle turned		+ 41.80	+112.49
	249°–37′			4325.18	7841.66
	180°–00′				
15–16 =	69°–37′	azimuth N. 69°–37′ E. 120.′0	(16)	+ 17.76	+113.62
	191°–30	angle turned			
	261°–07′				
	180				
16–17 =	81°–07′	= N. 81°–07′ E. 115.′0	(17)	4342.94	7955.28
	82°–00′			+ 57.41	— 17.43
	163°–07′				
	180				
17–14	343°–07′	= N. 16°–53′ W. 60.′0	(14)	4400.35	7937.85

By these calculations we are enabled to locate every point in the survey with its true coördinates and with respect to the original surface line 13–14.

If however the mine is entered by only one shaft the problem becomes even more difficult. It will be necessary under these conditions to drop two wires down the shaft, as far separated as possible, and placed in a manner similar to those in the former case

When the wires are resting quietly and precautions have been taken to see that no interference exists, the transitman on top from some known station locates by a course and distance each one of the wires thus making them part of the surface survey or else brings the wires into the azimuth of some known line. In the former case their coördinates may be calculated, and from these the azimuth of the line joining them determined. The distance

between the two wires is accurately measured and the transit then taken down the shaft to the station where the survey is to be carried into the level.

Two methods exist for "taking off" from the wires in the shaft, the more common of which will be mentioned later.

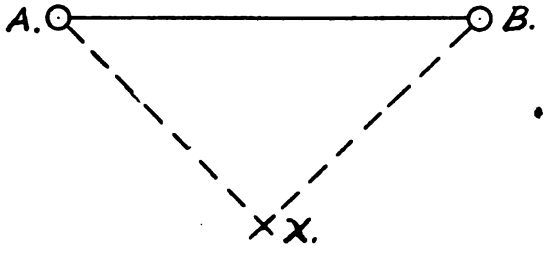

FIG. 3.

Having set up in the station at any convenient point X, Fig. 3 a horizontal distance is taken from the transit to each wire and the angle AXB is read. Thus three sides and an angle are known together with the azimuth of AB. This affords the means of determining the other angles and hence the azimuth of both AX and BX. If we sight on either A or B, therefore, with a zero vernier, we begin with a known azimuth and are able to run the survey as before into the level.

The more common method however is to set the transit up in the station, in a line which is the prolongation of the line AB, Fig.

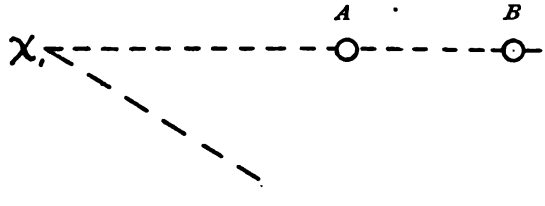

FIG. 4.

4. This is only accomplished by a series of trials. The man at the transit with an approximate set up sights over the telescope pointing first at one wire and then at the other, which are distinguished by candles held behind them. When the center of the instrument X after successive jogglings, is thought to be in line with the prolongation of AB, it is leveled up, and the telescope is then focused

upon the wires by moving the adjustable head till the line of sight *XAB* has been made to pass through *A* and *B* at the same time. This being the case, the transit is again leveled, the vernier made to read zero, and a sight taken on *AB*. *XAB*, the line of sight, hence has the same azimuth as the line *AB*.

The horizontal distance from the set up to each wire is taken, the difference giving the distance between. This however is also measured to check not only the calculation, but also the measurement of the same distance on top. Sighting on the wires as a back sight and locating any permanent station as previously described we are able to carry our survey from the azimuth of *AB* into the level and thus calculate the coördinates, by courses and distances of every point later located.

In the case of an inclined shaft it becomes necessary to use inclined readings, that is vertical angles. The transit will be set up as shown, Fig. 5, at some known point on the surface and a plumb

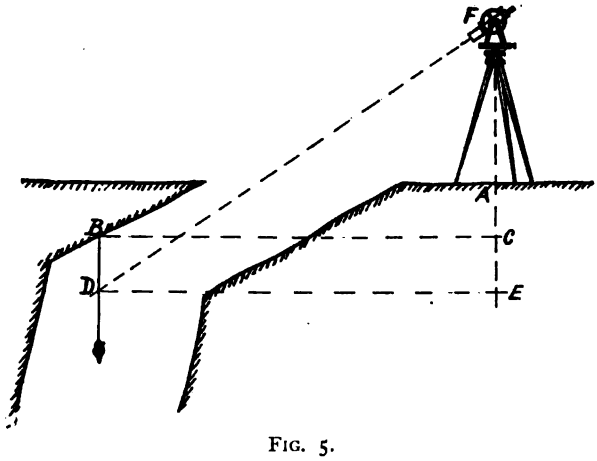

FIG. 5.

bob will be located, by azimuth, inclined angle and distance. This station just located is taken as a set up, a B.S. is read on the previous point, and other convenient points are determined in the incline. The horizontal distance *CB* between the points *A* and *B*, and also the vertical distance *AC*, is determined in the following manner. The elevation of *A* is known from the surface surveys which gives us the elevation of the H.I. The vertical angle *FDE* is read on the vertical arc of the transit and the inclined distance from the center of the horizontal axis (∴ vertically above *A*

at a distance of H.I.) to a point D on the plumb bob at a measured distance BD below B. With DF and the angle FDE known we may calculate the distances DE and EF.

DE is the horizontal distance between A and B, while $EF-(AF+EC)$ gives us the difference in elevation between A and B. This therefore locates B with respect to the surface surveys in both horizontal and vertical planes.

Special forms of transit may be required in this work, such as one with a top or side telescope, where it is possible to obtain a more nearly vertical line without interference from the horizontal plate than with the ordinary instrument. A mine transit of whatever kind should be as light as possible so as to be easy to carry up and down raises and for this reason they are often made of aluminum. Furthermore, weight is not required here as on the surface, to steady the instrument in the wind as but slight currents exist and these do not cause the transit to tremble.

Record books of all the surveys should be kept in the office so that instead of referring to the field note books, which are always somewhat more or less soiled, we may go directly to the more permanent office records where every detail of the original survey has been copied, in ink, together with the azimuth of each course, the coördinates and elevation of each station.

At the same time, where the surveys cover a number of years and the station numbers are numerous, it is well to have a "station number book" in which each station is recorded, the mine in which located and the ledger or reference book with page, where its coördinates may be found.

One purpose of all underground surveys is to map the workings that the development may be shown, the relation and position of the ore bodies on the various levels and that intelligent prospecting from the mine sheets may be possible. These maps are first a composite level map, which shows on one sheet all the developed workings on each level, each level being given a different color to distinguish one from the other. It is necessary also to have single level sheets, where only the workings on each individual level appear that the level may be studied more in detail. And finally a longitudinal section of the ore bodies from which the reserves and other data may be calculated.

[CONTRIBUTIONS FROM THE HAVEMEYER LABORATORIES OF COLUMBIA UNIVERSITY, No. 144.]

ON SOME ANALYTICAL PROPERTIES OF PINE WOOD OILS.

BY OTTO KRESS.

In view of the increasing importance of the oils derived from the distillation of pine wood * it is desirable to determine their leading analytical properties.

Four of these products, together with a sample of pure turpentine for comparison, have been subjected to examination by some of the methods most used in oil analysis with the results given below.

SAMPLES EXAMINED.

The samples examined were as follows:

Sample A. — A pure genuine turpentine, technically rated as of high quality, obtained through the courtesy of the Georgia Pine Turpentine Company.

Sample B. — "Turpyne No. 2." — A product of the Georgia Pine Turpentine Company resembling turpentine in its general properties and solvent power but having a characteristic odor.

Sample C. — "Turpyne X." — A product of the same company originally of high quality but which due to exposure had become slightly discolored and had increased in specific gravity from 0.858 to 0.8803.

Sample D. — "Harz Mountain Pine Oil." — A clear light yellow product, imported from Germany and used as a substitute for turpentine in the manufacture of varnish.

Sample E. — A heavy dark oil called "pine oil" but resembling rosin oil rather than any of the above in its physical properties. The sample was black, had a density of 1.02 + at 15° and a charteristic odor. It was said to have been freed from turpyne and from creosote.

Samples "D" and "E" had been in the laboratory from one to three years before the greater part of the work was done; while "A," "B" and "C" were examined within a few days after being received.

* On the technology of the pine wood oils see Clark, *School of Mines Quarterly*, 1888, 9: 162; and Walker, Wiggins and Smith, *Technology Quarterly*, 1905.

Comparison of Turpyne with Turpentine.

Specific Gravity and Index Refraction.

The specific gravities determined at 15° in a modified Geissler pyknometer * were as follows:

Turpentine	(Sample A)	$\frac{15°}{15°}$	0.8689
Turpyne No. 2	(Sample B)	$\frac{15°}{15°}$	0.8653
Turpyne X.	(Sample C)	$\frac{15°}{15°}$	0.8803

Determinations of refractive index by means of the Zeiss butyro-refractometer with sodium light showed:

Turpentine	(Sample A)	index at 25° 1.4676
Turpyne No. 2	(Sample B)	index at 25° 1.4725
Turpyne X.	(Sample C)	index at 25° 1.4701

Fractional Distillation of Sample B.

Two distillations were made to determine whether the fractions would vary in specific gravity or index of refraction, and whether any of these fractions would approach the properties of turpentine or of turpyne X.

In the first experiment 250 c.c. of the "Turpyne No. 2" were heated on a sand bath in an ordinary glass distilling flask and the distillate collected in nine fractions of 25 c.c. each and a tenth fraction of 17 c.c., a small amount of brownish viscous residue remaining in the flask at the end of the operation.

The temperature of distillation and refractive index of each fraction is shown in Table I.

Table I.

Results of First Fractional Distillation.

Number.	Distilled at	Index of Refraction at 25°.
I.	63°–160°†	1.4690
II.	160	1.4699
III.	160–164	1.4703
IV.	164–165	1.4704
V.	165–167	1.4710
VI.	167–169	1.4713
VII.	169–171	1.4722
VIII.	171–173	1.4731
IX.	173–178	1.4749
X.	178–190	1.4791

* The glass cap of the side tube was replaced by a small piece of rubber tubing carrying a glass tube drawn to a capillary opening at the top. This permitted expansion without appreciable evaporation.

† The first fraction contained a little water which separated mechanically before the index of refraction of the oily portion was determined.

The index of refraction was determined on the day following the distillation, the fraction being kept in stoppered bottles.

Comparison of the fraction shows a steady increase in the refractive index with increasing temperature of distillation. The first half of the sample distilling below 167° shows an average index of 1.4701, which is identical with that of Sample C (Turpyne X.).

No significant differences in the densities of the fractions were found.

The second distillation experiment was conducted in the same manner as the preceding, except that the first 12 c.c. of distillate, containing all of the water, were collected separately, this fraction being designated "A." Nine fractions of 25 c.c. each were then collected and distillation stopped. The residue after cooling was poured out of the flask and kept in a corked bottle under the same conditions as the distillates.

Each distillate was examined in the refractometer immediately after the completion of the distillation and again after three days. Since three days standing did not cause any appreciable increase in the readings, it is evident that the results obtained in the preceding test cannot have been appreciably affected by the fact that the samples stood over night before examination.

The results of the second distillation are shown in Table II.

TABLE II.

RESULTS OF SECOND FRACTIONAL DISTILLATION.

Fraction Number.	Distilled at	Index of Refraction at 25°.	
		At End of Distillation.	After 3 Days.
	° °		
A*	80–153	1.4683*	1.4682
I.	153–159	1.4695	1.4697
II.	159–161	1.4696	1.4699
III.	161–164	1.4702	1.4700
IV.	164–165	1.4702	1.4702
V.	165–167	1.4707	1.4708
VI.	167–169.5	1.4719	1.4718
VII.	169.5–172	1.4722	1.4728
VIII.	172–175	1.4742	1.4742
IX.	175–183	1.4776	1.4775
Residue.		†	

Here, as in the first distillation, the earlier fractions are well within the limits of variation of pure turpentine, and the average

* This portion contained some water which had apparently separated before the oil was examined in the refractometer.

† Index of refraction too high for the scale of the butyro-refractometer.

refractive index of fractions I. to V. is the same as that of Sample C ("Turpyne No. 2").

The indices of refraction of 42 samples of turpentine were determined by McGill in 1901.* Reducing the figures given by McGill to 25° by means of the correction factor employed in our work, they are as follows: Minimum, 1.4644; maximum, 1.4699; mean, 1.4671. The pure turpentine used in our experiments agreed closely with McGill's mean, and the "Turpyne X." had a refraction practically coinciding with the maximum found by McGill among his 42 samples of turpentine, while the "Turpyne No. 2" showed a refractive index considerably above McGill's maximum.

The earlier fractions of the distillate obtained from "Turpyne No. 2" were within the limits suggested by McGill for turpentine. McGill states that nine tenths of a sample of turpentine should distill below 180°. When nine tenths of the "Turpyne No. 2" had distilled the boiling point was 178°, or well within McGill's limit for genuine turpentine in this respect.

Influence of Temperature upon Specific Gravity.

The specific gravities of turpentine and the two turpynes at $\frac{15°}{15°}$ and $\frac{25°}{25°}$ were determined by means of the Geissler pyknometer with the following results:

	Turpentine.	Turpyne X.	Turpyne No. 2.
Specific gravity at $\frac{15}{15}$	0.8689	0.8803	0.8653
Specific gravity at $\frac{25}{25}$	0.8627	0.8749	0.8592

The coefficient of expansion is therefore nearly the same for the three samples.

Influence of Temperature upon Index of Refraction.

By means of the Zeiss butyro refractometer with the apparatus for regulating the temperature by means of a water current, the index of refraction of each sample was determined at various temperatures between 20.5° and 41°. Lag in the thermometer readings apparently affected the accuracy of individual observations, but the results as a whole showed that the effect of temperature

* Bulletin No. 79, Laboratory of the Inland Revenue Department, Ottawa.

upon index of refraction was approximately constant in each case within the limits given.

The influence of temperature, as observed in these requirements, was practically alike for the three samples. In each case the decrease of the index of refraction for each rise of 1° C. was 0.00045 to 0.00047.

Influence of Long Standing upon Index of Refraction.

Standing four months in well-filled, tightly stoppered vessels had no appreciable influence upon the index of refraction of any of the three samples. The fractions obtained in the second distillation of Sample B were kept in bottles only one third to one half full so that each portion was exposed to more than its own volume of air, and the refractive indices redetermined after fourteen days and again after four months storage in a dark closet. Averaging the results in order to eliminate as far as possible the effects of experimental errors the index of refraction was found to have increased only from 1.4742 to 1.4745 as a result of the four months storage.

Iodine Absorption Numbers.

Attempts were made to determine the iodine numbers of turpentine, Turpyne X., and fractions I. and IX. obtained in the second distillation of Turpyne No. 2 (see above). Both with the Wijs' and Hanus solutions Turpyne XX. showed distinctly higher iodine absorption than did the turpentine. The Hanus method showed results slightly lower for fraction I. and somewhat higher from fraction II., than for turpentine tested under the same conditions.

In all of these cases, however, the determinations were unsatisfactory, the end-point being uncertain because of rapid reappearance of the starch-iodine color, and the amount of halogen absorbed being much influenced by the amount present as will be seen from the results given below.

HALOGEN ABSORPTION AS AFFECTED BY AMOUNT OF REAGENT.

Turpentine and Turpyne X. were treated under similar conditions with varying amounts of iodine. In all cases the samples stood about one hour after adding the halogen solution before the excess of halogen was titrated by thiosulphate. In Table III. the results are arranged to show the effect of the amount of reagent

added. For each determination (or average of duplicate determinations) the ratio of iodine to oil is shown along with the iodine number found.

TABLE III.

INFLUENCE OF AMOUNT OF REAGENT UPON IODINE ABSORPTION OF TURPENTINE AND TURPYNE.

Hanus Method.

Turpentine. Ratio of Iodine to Oil (by Weight).	Iodine Number Found.	Turpyne. Ratio of Iodine to Oil (by Weight).	Iodine Number Found.
1.3 : 1	98.8	1.1 :	113.1
1.76 : 1	106.3	1.55 :	181.6
2.84 : 1	160.0	3.55 :	195.6
3.74 : 1	187.4	4.81 :	291.0
4.05 : 1	205.7	6.56 :	351.6
6.05 : 1	211.4	7.21 :	354.5
8.88 : 1	266.1		
9.4 : 1	271.2		

Wijs Method.

1.05 : 1	92.7	1.78 : 1	131.0
2.00 : 1	147.5	2.11 : 1	159.8
3.17 : 1	219.4	3.47 : 1	257.4
4.00 : 1	232.5	4.03 : 1	283.1
7.28 : 1	302.8	6.97 : 1	342.0
8.01 : 1	301.8 (?)	8.43 : 1	378.7
10.02 : 1	438.2 (?)	10.18 : 1	446.3

It is is evident that for each sample and with each method the result obtained varies enormously with the ratio of iodine to oil; the greater the amount of halogen present the greater the amount absorbed by a given weight of oil. In these determinations as well as those given below, there was constant trouble with the end-point in the final titration, the blue color reappearing quickly and repeatedly after the solution had been decolorized by thiosulphate, as if some of the iodine taken up by the sample was so loosely held as to split off as soon as the excess of reagent had been consumed by the thiosulphate. On account of this unavoidable source of error it is impossible to deduce any quantitative expression for the influence of excess of reagent upon the iodine number obtained.

The result of similar tests upon Sample D and E (see description of samples above) are shown in Table IV. In these determinations the Hanus solution was used and the time of standing was one hour.

TABLE IV.

INFLUENCE OF AMOUNT OF REAGENT UPON IODINE ABSORPTION OF SAMPLES D AND E.

Hanus Method.

Sample D. Ratio of Iodine to Oil.	Iodine Number Found.	Sample E. Ratio of Iodine to Oil.	Iodine Number Found.
1.10 : 1	69.0	1.08 : 1	54.2
1.85 : 1	101.1	2.10 : 1	75.0
2.07 : 1	116.8	3.88 : 1	96.8
2.75 : 1	121.0	4.10 : 1	115.8
3.02 : 1	123.0	6.50 : 1	126.5
3.13 : 1	125.6	7.07 : 1	168.0
3.70 : 1	126.4	8.27 : 1	172.8
3.86 : 1	127.9		
7.42 : 1	135.8		

The repeated reappearance of a blue color immediately after the end-point had apparently been reached in titrating the excess of halogen by thiosulphate has been mentioned in discussing the results given in Table IV. This occurred with all of the samples examined. It had also been noted in previous work by von Sholly in this laboratory. On account of the uncertainties this introduced no attempt will be made to compare the results obtained by the different methods or to express quantitatively the influence of the amount of halogen present upon the amount absorbed. It is to be noted, however, that even when the amount of reagent was insufficient some halogen always remained unabsorbed; while with increasing amounts of reagent the absorption figures continued to increase as far as it was practicable to carry the test. Evidently, therefore, the halogen absorption methods as used satisfactorily in fat analysis are of comparatively little value in the examination of pine wood oil, and if applied to mixtures containing such oils without a knowledge of the influences which affect the results, may easily lead to erroneous conclusions.

SUMMARY.

The light oil obtained by the distillation of pine wood and resembling turpentine in its solvent properties resembled it also in the cases here studied, in specific gravity and in rate of expansion on warming. The influence of temperature upon the index of refraction was also alike for turpentine and for the pine wood distillate, but at any given temperature the refractive index of the latter was higher than that of the former; in fact, the wood oil showed

a higher refraction than the maximum limit allowed for turpentine by McGill.

A sample of pine oil which had been treated for the removal of its characteristic odor showed a density and an index of refraction each practically coinciding with the maximum found in genuine turpentine by McGill. Like the first sample it resembled turpentine in the rate of change of density and refraction on warming.

When a portion of the first sample was submitted to fractional distillation, the indices of refraction of successive portions of distillate were found to increase with the temperature of distillation. The densities of all the refractive indices of the earlier fractions were within the limit of variation of turpentine. The portion distilling at 169°–172° showed the index of refraction of the original sample. The index of refraction of the distillate rose much more radidly in the later than in the earlier part of the distillations.

The turpentine and the two samples of "turpynes" were kept four months in well-filled closed vessels in a laboratory closet without appreciable change in density or index of refraction. The average refractive index for 8 distillates kept in bottles one third to one half full in the same closet rose during the four months from 1.4742 to 1.4745.

The index of refraction of turpentine and the light oils distilled from pine wood is thus fairly constant if reasonable care is taken to protect the samples from light, air and moisture. The influence of temperature upon the refractive power is somewhat greater for these than for fatty oils; a rise of 1° causing in our experiments a decrease of 0.00045 to 0.00047 in the index of refraction.

The halogen absorption methods were tried in different modifications, but gave exceedingly variable and uncertain results with all of the pine wood oils. The end-point of the final titration was unsatisfactory and the amount of halogen absorbed was greatly influenced by the excess of reagent added.

MALLEABLE CAST IRON.

By BRADLEY STOUGHTON.

Malleable cast iron is iron which, when first made, is cast in the condition of cast iron, and is made malleable by subsequent treatment without fusion. Gray cast iron is weak and brittle, on account of its flakes of graphite which destroy its continuity and form planes of easy yielding. It will readily be understood that if the amount of graphite were less, or if its flakes occurred in a very finely pulverized form, or both, the material would be stronger and would endure a slight degree of deformation without cracking; that is, it would be malleable. These changes are indeed brought about in the manufacture of malleable cast iron. The process, which was invented by Réaumur in 1722, but has only been in practical use about 100 years and of importance less than 50, is a very ingenious operation. Malleable castings have two of the greatest advantages of cast iron; namely, fluidity and a low melting-point, combined with about three fourths the strength and one sixth the ductility of steel. They are very popular for railroad rolling-stock construction, especially for draw-bars, couplers and knuckles, on account of their high resiliency, resistance to shocks, and ability to be made into thin, light castings. Probably about one half of the malleable cast iron production of the United States goes into railroad work, although it is to be observed that at present the use of steel castings for this purpose is increasing relatively faster. The next most important use is for pipe-fittings, where malleable cast iron is equally advantageous; also for small machinery castings needing to be strong andlight, household, harness and building hardware, etc.

Process. — Pig iron of the proper kind is first melted and cast into molds of the desired size and shape. In these operations two precautions are observed: first, the proportion of silicon is low, and second, the castings are not allowed to cool too slowly. Precipitation of graphite is a slow action and does not occur unless ample time is allowed or there is sufficient silicon to produce a strong chemical action. In the intentional absence of these factors malleable castings, as first made, are practically free from graphite and consist entirely of white cast iron, — hard, brittle and weak.

After cooling the castings are cleaned and packed in some pulverized material, as iron ore, mill scale, lime, sand, placed in an annealing furnace and heated to a temperature of 675° to 725° C. (1250° to 1340° F.), which is, roughly, 450° C. below their melting-point, and at which temperature they are kept for many hours. While under this heat there occurs the precipitation of graphite, which normally should have occurred during solidification, or shortly thereafter, and in the majority of cases almost all the combined carbon throughout the body of the casting is changed to graphite. But the graphite does not here form in flakes, as in ordinary gray cast iron, but in a finely comminuted condition, like a powder, to which the name of "temper carbon" or "temper graphite" is given. In this form it is not nearly so weakening or embrittling to the casting as flakes of graphite would be.*

Properties of Malleable Cast Iron. — Malleable cast iron consists almost entirely of ferrite and temper carbon. It has a tensile strength of 40,000 to 60,000 pounds per square inch, which is about double that of gray cast iron, with an elongation of 2½ to 5½ per cent. in two inches and a reduction of area of 2½ to 8 per cent.† A one-inch square bar on supports 12 inches apart should bear a load at the center of at least 3,500 pounds, and be deflected at least half an inch before breaking. Thin sections should be capable of flattening out under a hammer and bending double without cracking.

Total Carbon in Malleable Cast Iron. — In melting the iron we take pains to produce a low total carbon, and if we anneal in iron ore or mill scale the carbon is still further reduced by a curious reaction which takes place between it and the iron oxide —

$$3C + Fe_2O_3 = 3CO + 2Fe,$$

whereby it forms carbon monoxide and is eliminated. Sometimes, when the sections of metal are thin, we may eliminate almost all the carbon to the very center of the casting, which makes a more ductile material. It must be observed, however, that this reduction of carbon is not an essential feature of annealing and that the real function of this operation is to change the combined carbon

* We may liken this to two samples of putty, in one of which had been embedded a large number of plates of mica, and in the other the same amount of mica ground to powder.

† In the case of iron very carefully melted and annealed in iron oxide, the elongation may go as high as 8 per cent., and the reduction of area as high as 12 per cent.

to temper carbon, for the malleable cast iron owes its superiority over gray cast iron chiefly to the finely pulverized form of its temper carbon.

Melting in the Air-Furnace. — The commonest melting-furnace for malleable cast iron is the air-furnace, because thereby we get a better control of the metal than in the cupola and the ability to produce castings with lower total carbon, lower sulphur and any desired amount of silicon. After the metal is melted it is retained in this furnace for 15 minutes to an hour longer, and test samples are taken at intervals, from the fracture of which and the temperature of the iron we determine the correct moment for tapping. The fracture of the test ingot sample should be a clear white throughout, except when the castings are to be of very light section, in which case the metal might be tapped when the test sample shows a few specks of graphite in the center. The practice of judging from test samples is different in each foundry, but there must be some system which insures that the metal shall be of such a composition when tapped that the castings will have not more than a trace of graphite, if any at all (say, less than 0.15 per per cent. in small castings and a little more in larger ones). It is to be remembered that any graphite is a detriment. The longer in the furnace after melting the more silicon will be burned out, and therefore the greater the liability to graphite. Consequently, the hotter we want to get the iron the higher must be the percentage of silicon in it to start with. Also, the longer the metal is in the furnace after melting the more carbon will be burned out of it.

Tapping the Air-Furnace. — When the time comes to tap the furnace we may either allow a small stream to flow, which is caught in ladles and immediately poured into the castings, whereby it takes from 20 minutes to an hour to empty a furnace of 10 to 30 tons capacity, or else we may allow the metal to run out in a big stream into a large ladle, from which it is repoured into smaller ones for casting. The first method gives a less uniform product, because the last part of the bath, having been exposed longer to the oxidizing influence of the furnace, is lower in silicon and total carbon than the first part. This is not altogether a disadvantage, however, because the first metal, having come from the top of the bath nearer the flame, is hotter, and therefore very suitable to pour into smaller castings; and these smaller castings, because

they cool more rapidly, can well contain more silicon without danger of graphite precipitating. A disadvantage of pouring the metal first into a big ladle is that it must be hotter when it comes from the furnace, and moreover the first ladle must be preheated. A middle course is possible: We may pour the top of the bath into small castings by means of ladles receiving their metal direct from the furnace tap-hole, and then enlarge the tap-hole and take all the rest of the bath into a big ladle, whence it can be poured into other ladles and go to the larger-sized castings.

Cupola Melting. — In cupola melting we get metal having practically the same composition at all times of the heat,* and also about the same temperature. It is also cheaper in fuel, and especially so when it is desired to get very fluid iron, because obtaining hot iron in the air-furnace requires a continuation of the heating after the iron is melted, and this entails not only the use of more fuel, but also a burning out of silicon and carbon, both of which elements increase the fluidity of the metal.† Cupola metal is higher in total carbon and in sulphur, both of which decrease strength and ductility. Because of the higher total carbon it is more difficult to prevent graphite separating; therefore, cupola metal is used for light castings, which cool more quickly and do not usually require so much strength.

Regenerative open-hearth furnaces are also used in melting iron for malleable castings.

Annealing-Boxes. — After the castings are cooled they are carefully cleaned from all adhering sand by tumbling them around in a tumbling-barrel (in which they are mixed with star-shaped pieces of metal something like children's jackstones), or by sand-blast, or by some other suitable method. They are then packed in the cast-iron "saggers" or annealing pots, or boxes, together with the packing. Sometimes, though rarely, the tops of the saggers are closed by means of an iron cover, sometimes by a thick layer of the packing in the upper part, and sometimes by clay or wheel-swarf.

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* Except for the slightly higher sulphur at the beginning and end.

† By "hot iron," here, we mean "fluid iron," *i. e.*, the degree of heat above the melting-point; and in this sense hotness includes both the temperature and the state of impurity. An iron with 2 per cent. total carbon and 0.60 per cent silicon at 1300° C. (2375° F.) will not be nearly as fluid or as far above its melting-point as one with 3 per cent. total carbon and 0.75 per cent. silicon at the same temperature.

Annealing-Ovens. — The boxes are then placed in the annealing-ovens in such a way that the flame may play around them as completely as possible. The flame usually comes in at the top of the ore and goes out at the bottom along the sides, and, thence through flues underneath. The fuel used may be coke, coal, oil or gas, the two latter being preferable on account of the better control of the temperature, which should be increased at a very gradual and uniform rate during the heating up, and kept as constant as possible during the annealing period.

Annealing Practice. — It takes about six days for the annealing operation, including heating up and cooling down. Sometimes this can be shortened a little by decreasing the time at the full annealing heat, or by cooling rapidly or by drawing the saggers out of the oven and dumping them while the contents are still at a dull-red heat. This practice is not conducive to a good quality of castings and should never be permitted in important cases. The time at the full heat should never be less than 60 hours, and preferably it should be more than that. If less, the temperature of annealing must be higher, and this decreases the strength and ductility of the castings. Annealing should not occupy too long a time, however, unless the temperature is quite low, because the temper carbon tends to draw together into larger flakes; besides which the metal may become oxidized between the grains, or "burnt."

Packing. — As originally carried out the castings were annealed in a packing of iron oxide crushed to a size less than a quarter of an inch in diameter. The packing must surround the castings at every place, both inside and out, and no two castings must touch. Iron ore, mill scale, "bull-dog," and similar forms of iron oxide are used for this purpose. Usually two or three parts of old packing are used with one part of new packing, because all new packing is too energetic in its chemical action on the carbon, and all old packing will not be energetic enough in decarburizing the surface of the castings.

Annealing in iron oxide produces a white skin where the casting has been deprived of its carbon, and a black interior, due to the temper carbon, whence the name of "black heart malleable" for this material. Tests have shown that the casting with this white skin upon it is much stronger than a similar one which has not been decarburized on the surface, and therefore the packing in

iron oxide is advantageous, even though not an essential feature of the operation. When, according to recent innovations, adapted for the sake of cheapness, the castings are packed in some non-oxidizing material, such as sand, clay or lime, they may receive as perfect an annealing, as far as the production of temper carbon is concerned, but will be without the white skin and of lower strength.

Composition of Iron Used. — The pig iron employed in this process is sold under the name of "malleable Bessemer." As it is the composition of the metal poured into the molds which determines the success of the annealing and the quality of the product, we shall consider first the kind of metal needed there, and from that calculate the composition of the mixture necessary to charge into the air-furnace or cupola. In the air-furnace we do not judge the iron by its chemical analysis, but by the appearance of the test-ingot fracture when we are ready to pour, but as the second quality depends upon the first, it is the same thing in the end.

Silicon. — The proportion of silicon will depend upon the size of the casting and the amount of total carbon, because the greater each of these is the less will be the amount of silicon that will cause a precipitation. It might at first appear then that the less silicon the better; but this is not altogether so, because temper carbon will not come out during annealing unless a certain amount of silicon is present; and the more there is, the more quickly, easily, and completely will the precipitation occur. For castings one inch thick the silicon may be as low as 0.35 per cent., but this is unusual, as three quarters of an inch thickness is rarely exceeded. For half-inch castings the silicon will be about 0.60 per cent., and for very thin and light castings with low total carbon and high sulphur (say 0.205 to 0.300 per cent.), the silicon may be up to 1 per cent. To the percentage of silicon desired in the castings we must add the amount which will be burned out in melting. In the cupola this will be about 0.20 to 0.25 per cent., and in the air-furnace from 0.15 to 0.50 per cent., or more if desired, depending on the length of time the metal is kept in the furnace after melting. The hotter we want the iron or the more total carbon we desire to burn out, the longer this time must be, and therefore the higher the silicon in the original mixture charged.

Sulphur. — Sulphur increases the tendency of castings to check, which is especially important in malleable work on account of the shrinkage of white iron being nearly double that of gray iron.

Sulphur also reduces the strength and the ease of annealing. For this reason over 0.06 per cent. should not be permitted in castings requiring strength, but it actually runs up to 0.200 and 0.300 per cent. in inferior metal, both in America and England, and especially in small castings, which do not need strength so much, and which, having less length for shrinkage, are not so liable to be checked by cooling strains.

Manganese. — Low manganese is preferred by many foundries, and one of the highest authorities in America places the limit at 0.20 per cent. It should be remembered, however, that the manganese should be at least twice the sulphur, and preferably three times, though not when the sulphur is as high as 0.300 per cent. Manganese of 0.50 per cent. tends to decrease checking. It also protects silicon from oxidation, both during melting and annealing, and on this account hastens and makes more complete the precipitation of temper carbon. It also protects the iron itself from oxidation during annealing and thus prevents the formation of "scaled" castings. More than 0.60 per cent. manganese makes the iron hard and difficult to machine, which is disadvantageous, especially for pipe-fittings, which must be threaded with economy in order to meet the trade competition.

Phosphorus. — Phosphorus makes the metal fluid, which is especially desirable where total carbon and silicon are low, or where sulphur and manganese are high. On the other hand, it diminishes two of the most valuable properties of the material: its resiliency and resistance to shocks. It also makes the metal hard, difficult to machine and liable to check, and amounts over 0.225 per cent. should never be permitted by engineers where the castings are subjected to strain.

Total Carbon. — Total carbon below 2.75 per cent. gives trouble in annealing and therefore makes the castings weak. It also makes the metal more sluggish. It is difficult to get as low as this in cupola melting, although mixing in large percentages of steel scrap and allowing the metal to run out of the cupola as fast as melted will reduce the proportion appreciably. In air-furnace practice the total carbon may be reduced as far as necessary, but with the disadvantages mentioned above. Annealing in iron oxides also removes carbon from the outer layers and even to the very center of thin castings. The lower the total carbon in the *annealed* castings, the better.

Scrap Used. — Not more than about 20 per cent. of bought scrap is used on the average in American practice, and a good deal of this is steel, on account of the desirability of lower total carbon, and because iron scrap is too impure, too variable and too uncertain in sampling and chemical analysis for castings requiring strength, such as those for railroads and machinery. There is, however, in addition always a large amount of "return scrap" from the foundry, consisting of defective castings, sprues, gates, etc., which latter, in the case of small castings, may be greater in weight than the castings themselves. This return scrap is low in total carbon and silicon as a result of having already suffered the melting changes.

Shrinkage. — The shrinkage of malleable iron from casting is almost as great as that of steel, because almost no graphite forms. The amount of silicon and the sectional area of the castings are still the determining factors in this connection. Indeed, by means of the measurement of the section and the percentage of silicon we may estimate the shrinkage, or by means of the section and the shrinkage we may estimate the silicon very closely, other conditions and impurities being normal. The following table gives the necessary data for these estimations:

SHRINKAGE IN INCHES PER FOOT OF LENGTH.

Percentage of Silicon.	¼ Inch Square.	½ Inch Square.	¾ Inch Square.	1 Inch Square.
0.35	0.225	0.200	0.190	0.175
0.50	0.220	0.195	0.183	0.170
0.75	0.215	0.190	0.176	0.162
1.00	0.211	0.183	0.137	0.102

Expansion Due to Temper Carbon. — It is a very interesting fact that when malleable cast iron is annealed and the temper carbon precipitates, the casting expands to a size approximately equal to that which it would have had if the graphite had separated during solidification and gray cast iron had been produced in the first instance. In other words, the temper carbon, although in a very finely powdered condition, occupies about the same amount of space as an equal weight of graphite, and causes about the same ultimate difference in size between the original pattern and the annealed casting as when gray cast iron is made.

Miscellaneous Iron Products. — In the form of small castings

malleable cast iron and similar products often appear under the name of steel, because under that name the producer finds a readier market for them. On account of their fluidity they may be cast very cheaply in small sizes, and therefore the temptation to use them as a material for "cast-steel hammers," * "hard-steel" bevel gears, "semi-steel castings," and even automobile "steel" drop-forgings, is a strong one. Engineers are warned to be on their guard against a deception of this kind, for legal redress has been sought many times in vain. A clever lawyer may easily confuse and outwit a judge or jury with the involved definitions and technical descriptions necessary to make the distinction clear. It is usual for the manufacturer when putting material of this kind upon the market to qualify the name "steel" with some other letters or name, such as "P. Q. steel," "Smith steel," etc.; but they all differ from true steel in that they were not "cast into an initially malleable mass." Some are made by melting a large proportion of steel with cast iron, after which the cooled metal may or may not be annealed in iron oxide. Others are made by a long or thorough annealing of ordinary malleable castings in iron oxide, by means of which the metal is decarburized to some depth, and is then carburized again by a cementation process. This makes a very good material for some purposes, such as small bevel gears not requiring strength or much ductility, but it ought not to be called "steel." Indeed there are some purposes for which malleable iron castings are even better than steel castings. If the purpose for which it is to be used does not require any other properties than malleable cast iron possesses, then it should be used under its true name, but if it is to be used under circumstances where it is liable to strain, calling it "steel" will not enable it to stand up under the work any better. The confusion is the more easy because genuine steel is made by the cementation of wrought iron, and wrought iron goes in England under the name of malleable iron. In America we seldom call wrought iron "malleable iron," but we often abbreviate malleable cast iron to "malleable iron," or even to "malleable."

* The trade would ordinarily understand by this name hammers made of crucible steel, so the use of this name is really a fraud.

THE ELECTROLYTIC TREATMENT OF GALENA.*

By EDWARD F. KERN and HERBERT S. AUERBACH.

In reviewing the literature on the metallurgical treatment of lead sulfide ores, one will find that considerable work has been done in attempting to overcome the difficulties which are encountered, especially in roasting the ore preliminary to the smelting. The general method of treating this class of ore, at present, is as follows: The sulfides are first converted into oxides, as far as possible, by some roasting process, and then the oxides are reduced to metal by means of charcoal or coke, either in a reverberatory or a blast furnace, thereby obtaining metallic lead and slag. Another method of treating the ore is the "Roasting" or "Reaction Process," which is subjecting the raw ore to an oxidizing atmosphere in a reverberatory furnace at a temperature of 450° to 600° C., forming lead oxide and lead sulfate. No attempt is made to oxidize all the sulfide, as then the object of the process, which is the interaction of lead sulfides with lead oxide and lead sulfate, would be hindered, besides much difficulty and additional expense would be encountered. Having partially oxidized the ore, the temperature of the furnace is raised to about a light yellow heat, causing the oxides and sulfate of lead to react with the unoxidized lead sulfide, producing metallic lead. Whether the ore be treated by either of these methods, the extraction of the lead, as metal, involves considerable loss of heat, and loss of metals by volatilization and in the waste products, also the evolution of volumes of obnoxious SO_2 fumes, besides the "wear and tear" on the furnaces is no small item.

For a matter of comparison, some of the disadvantages of smelting may be tabulated as follows:

A. In the roasting furnace.

1. Fuel required for heating.

*This article is based on an investigation, conducted by Mr. H. S. Auerbach during 1906, in the Metallurgical Laboratory of the School of Mines of Columbia University in the City of New York.

2. Loss of heat by radiation.
3. Loss of metal by volatilization, and as flue dust.
4. Expense of maintaining furnace.
5. Requires experienced and high-priced labor.
6. Incomplete oxidation of the ore.
7. Loss of sulphur content of ore.
8. Evolution of obnoxious SO_2 fumes.

B. In the reduction furnace.

1. Maintenance of furnace.
2. Values lost in slag.
3. Values lost as flue dust.
4. Loss of sulfur content of ore.
5. Excessive loss of heat by radiation and up the stack.
6. Evolution of obnoxious SO_2 fumes.

There are, no doubt, other disadvantages which the practical smelter may add to these, however, for the present, one is forced to admit that, even if the loss of metal by volatalization could be reduced, and the elimination of SO_2 fumes could be suppressed, the principal economic objections would be overcome. As electricity is the latest agent to be introduced into the field of metallurgy, and by its aid a number of complexing problems have been solved, the question which presents itself is : cannot electricity aid us in the recovery of metallic lead from galena ? We know that within the past few years electricity has made great conquests in the production of metallic sodium, aluminium, copper, iron and some of the rarer metals, so when this research was undertaken we had in mind the invention of a suitable system of electrolysis which would eliminate some of the perplexities encountered in smelting galena.

Some of the advantages of an electrolytic method of treating lead sulfide ore, which were anticipated, are as follows :

1. Elimination of roasting the ore.
2. Saving of sulfur as a by-product.
3. Elimination of the SO_2 fume nuisance.
4. Loss of metals in slag, and by volatilization decreased.
5. Less flux required.
6. No blowing apparatus required.
7. No water for furnace jackets (important in arid regions).
8. Saving in fuel, in case no water supply.
9. No fuel required, in case of water supply (electric generation).

10. Cheaper plant and apparatus.

11. Can work in smaller units, or in larger units, depending upon ore supply.

12. Less labor required.

Although all of the above advantages may not be realized, they go far in stimulating investigation of electrolysis of this class of ore.

In reviewing the literature on the electric treatment of galena, it was found that very little had been published on the subject. The only commercial process recorded, at the time of undertaking the investigation (September, 1905) was the "Salome Process" (*Electrochemical Industry*, Vol. I., 1902, pp. 18–488; SCHOOL OF MINES QUARTERLY, XXIII., 1902, p. 272). By this process spongy lead is produced by placing crushed galena on a lead cathode in an electrolyte of dilute sulfuric acid. The reaction which occurs is the electrolytic decomposition of the dilute acid electrolyte, liberating oxygen at the anode, and hydrogen at the cathode. The liberated hydrogen in turn reducing the galena, producing spongy metallic lead and hydrogen sulfide; thus,

Electrolysis of electrolyte; $H_2SO_4 = \overset{(+)}{H_2} + \overset{(-)}{SO_4}$.

Secondary reaction at anode; $H_2O + \overset{(-)}{SO_4} = H_2SO_4 + \widehat{O}$.

Secondary reaction at cathode; $PbS + \overset{(+)}{H_2} = Pb + \widehat{H_2S}$.

This process was formerly worked on a commercially experimental scale, but it is stated that it was finally abandoned, due to the difficulty of obtaining pure galena, to the cost of the operation, and the difficulty of complete reduction of all the galena. As the product was intended to be used in the manufacture of storage-cell plates, the presence of a small amount of sulfur in the spongy lead proved a detriment. It was also proposed to utilize the liberated hydrogen sulfide in the manufacture of sulfuric acid by simple ignition, and to oxidize the spongy lead in the reverberatory furnace producing litharge, but the demand for this latter product was such as to discourage the continuance of the enterprise.

Another process, which appeared later, somewhat similar in principal to the Salome process, is Townsend's "Method of Electrolytic Reduction of Galena." This process is covered by Letters Patent, No. 815,881, March 20, 1906. It differs from the Salome process in that the electrolyte is a fused bath of a mixture of sodium chloride and sodium sulfide, and in that the galena is re-

duced by metallic sodium which is liberated at the cathode. The cathode is a pool of fused lead on which the galena floats, by reason of the difference in specific gravities of the electrolyte, the galena and the fused lead. The anode is either carbon or Acheson graphite. The cell which contains the electrolyte is made of cast iron, the sides of which are insulated with a refractory non-conducting material, such as silica or fire-clay bricks. The reactions which occur during the electrolysis are:

$$\text{Electrolyte: } (NaCl + Na_2S) = NaCl + \overset{(+)}{Na_2} + \overset{(-)}{S}.$$

$$\text{Secondary reaction at cathode: } PbS + \overset{+}{Na_2} = Pb + Na_2S.$$

The electrolyte and lead cathode are kept molten by being placed in a furnace and by reason of the high current density used.

Another patent, quite similar to that of Mr. C. P. Townsend, is the Valentine-Betts "Electrolytic Process of Smelting Lead Sulfide Ores," and is covered by Letters Patent, No. 816,764, April, 1906. This process is identical with the Townsend process, except that the galena ore is first ridded of its associated gangue by fusing it with a suitable flux in an ordinary reverberatory melting furnace. The lead is then recovered from the resulting matte by bringing it in contact with a cathode in a fused electrolyte of a mixture of alkali chlorides and sulfides. The advantage claimed to be gained by fusing the galena ore, preliminary to the electrolysis, is the removal of the gangue. The fluxes used are iron ore and limestone, in such proportions as to form a fusible slag with the silicious gangue. The resulting slags containing from 5 to 15 per cent. of lead, are resmelted in an ordinary smelting furnace for the recovery of its lead content. The second advantage claimed for this process is that by having the lead matte, in a fused condition, on the cathode, the conditions of electrolysis can be so regulated that only the lead of the matte is reduced, leaving the iron sulfide constituent as such, which is tapped from the cell when most of its associated lead sulfide is reduced. As this iron matte contains a small amount of lead, it must also be smelted in the ordinary manner (*Electrochemical and Metallurgical Industry*, May, 1906).

Experimental Part.

Before presenting the results of the investigation, it might be well to enumerate some of the properties which a suitable fused

electrolyte should possess, so that the reader may have in mind what directed the course of the research.

A salt, in order to be a suitable electrolyte, should possess the following properties:

1. A low melting point.
2. Stable at temperatures far above its melting point.
3. Have low vapor tension; not very volatile.
4. A good conductor of electricity.
5. Very fluid, and homogeneous.
6. A low dissolving power for the metal, which is reduced at the cathode.
7. The anion should be one which has little, or no, corroding action on the insoluble anode.
8. It should have little or no corroding action on the cell.
9. A low "voltage of decomposition" (heat of formation).

The first determinations made were on the freezing points of a series of single salts, and mixtures of two or more salts. Note was also taken of the temperature of volatilization.

The apparatus, which was used for measuring the temperatures, was a Le Chatelier thermo-electric pyrometer,* using a couple of pure platinum wire and platinum containing about 10 per cent. iridium. It was calibrated, from time to time, by noting the degree of deflection produced by placing the couple in substances of known boiling, or freezing points. Those used were:

Boiling	point	of distilled water	= 100° C.
"	"	" naphthalene	= 218° C.
Freezing	"	" lead	= 327° C.
"	"	" zinc	= 420° C.
Boiling	"	" sulfur	= 445° C.
Freezing	"	" antimony	= 622° C.
"	"	"aluminium	= 655° C.
"	"	" silver	= 955° C.
"	"	" copper	= 1083° C.

The water, naphthalene and sulfur were boiled in test-tubes and the junction of the thermo-electric couple was held in their vapors. The metals were placed in small French clay crucibles, covered with powdered charcoal and heated in a Fletcher dental gas-blast

*Le Chatelier and Boudouard "High Temperature Measurements," American ranslation by Burgess, 1904, p. 120, p. 170.

furnace. In order to protect the couple from alloying with the molten metal, and to protect it from the injurious vapors and gases, it was inclosed in an unglazed porcelain tube, and the tube inserted in the crucible.

Having obtained the deflection of the galvanometer for the above standard temperatures, a standard curve was plotted, by using the deflections as abscissa and the temperatures as ordinates. By passing a line through these points on cross-section paper, the temperature curve was made. In making the curve the temperatures plotted were "corrected temperatures," obtained by subtracting the temperature of the cold junction from that of the hot junction, thus giving the true temperature which produced the deflection. When using the couple for determining the freezing point, or volatilization point, of a salt, the deflection of the galvanometer was noted, then by referring to the temperature curve, the temperature corresponding to the deflection was obtained, then by adding to this the temperature of the cold junction, the "corrected temperature" was given.

The determination of the freezing points and the volatilization points of the salts were made in the same manner as when calibrating the couple, that is by placing the salt in a small crucible, fusing it in a small Fletcher dental gas-blast furnace, and then inserting the couple in the fused mass. Clay crucibles were used for chlorides, graphite crucibles for fluorides, and in some cases when extremely high temperatures were had, the fusions were made in a platinum crucible. In order to prevent the fused salts from corroding the thermo-electric couple, it was protected by tubes. The tubes used for this purpose for the chlorides were unglazed porcelain, and for the fluorides were thin tubes made of Acheson graphite, the couple being insulated by strips of mica.

It was noticed that at the moment of solidification of a fused salt the temperature may vary in different parts of the crucible, *i. e.*, near the edge of the crucible the mass may be hotter or colder than at the center; or near the top it may be hotter or colder than at the bottom. In order to guard against discrepancies from such sources the couple was always placed in the center of the molten mass of salts.

It was also observed that the fused salt froze in one of the following ways:

1. Crystals bridging from the sides of the crucible towards the

center, until the surface was covered with crystals, then a gradual solidification of the remaining salt.

2. Gradual solidification of the mass from the center, the crystals radiating from the pyrometer tube and growing throughout the molten mass.

3. Instantaneous change throughout the mass, from liquid to solid state.

4. Separation of the mixed salts into two distinct layers, one of the layers freezing before the other.

FREEZING POINT OF SALTS AND MIXTURES OF SALTS.

Cryolite. — F.P. = 990° C. (Heated to about 1200° C. and let cool.) This same material was reheated several times to about 1400° C. and allowed to cool, giving a final F.P. of 935° C. During the heating, fumes were expelled. This latter F.P. remained fairly constant.

A sample of cryolite from a different source was melted and allowed to cool. Its F.P. was 980° C. It was reheated in the same manner as the first sample, several times to about 1400° C. During the reheatings some fumes were given off. The final F. P. was 957° C.

This dropping of the freezing point, and finally becoming approximately constant, indicates that a constituent of the material was expelled at the high temperatures to which it was heated.

Fluorspar. — Heated to 1200° C. but could not fuse.

Sodium Fluoride. — F.P. = 965° C. Did not change appreciably on remelting.

Potassium Fluoride. — F.P. = about 263° C. No very sharp retardation of galvanometer needle. Mass froze gradually.

Sodium Chloride. — F.P. = 792° C. The salt decrepitates during heating, and is quite volatile, even below its melting point. Fumes given off when temperature of about 750° C. is reached.

Magnesium Chloride. — F.P. = 158° C.–160° C. Mass froze gradually.

Mixtures of Cryolite and Sodium Chloride. —

20 parts cryolite + 2 parts NaCl, F.P. = 955° C.
20 " " + 5 " NaCl, F.P. = 905° C.
20 " " + 10 " NaCl, F.P. = 862° C.

In each case fumes of NaCl expelled, but after mass once melted, less fumes than during first melting.

Mixtures of Cryolite and Fluorspar. —

20 parts cryolite + 2 parts Fluorspar, F.P. = 960° C.
20 " " + 5 " " F.P. = 895° C.
20 " " + 10 " " F.P. = 910° C.

Mixtures of Cryolite and Potassium Fluoride. —

20 parts cryolite + 2 parts KF, F.P. = 955° C.
20 " " + 5 " KF, F.P. = 920° C.
20 " " + 10 " KF, F.P. = 903° C.

Mixtures of Cryolite and Sodium Fluoride. —

20 parts cryolite + 2 parts NaF, F.P. = 956° C.
20 " " + 5 " NaF, F.P. = 910° C.
20 " " + 10 " NaF, F.P. = 877° C.

Mixtures of Cryolite and Potassium Chloride. —

20 parts cryolite + 2 parts KCl, F.P. = about 940° C.
20 " " + 5 " KCl, F.P. = " 909° C.
20 " " + 10 " KCl, F.P. = " 900° C.

This mixture settled out in two layers, so the latter results are not very trustworthy.

Lead Fluoride. — F.P. = 552° C.

Lead Chloride. — F.P. = 494° C. Started to volatilize at about 510° C., and at 697° C. large volumes of fumes, but mass not boiling.

Potassium Chloride. — F.P. = 769° C. Volatile above melting point.

Calcium Chloride. — F.P. = 772° C. Not very volatile even at higher temperatures.

Mixtures of Sodium Chloride and Lead Chloride. —

20 parts NaCl + 2 parts NaCl, F.P. = 770° C.
20 " NaCl + 5 " $PbCl_2$, F.P. = 761° C.
20 " NaCl + 10 " $PbCl_2$, F.P. = 723° C.

When heated above melting point, fumes given off; the more fumes, the higher the temperature.

Mixtures of Potassium Chloride and Lead Chloride. —

20 parts KCl + 2 parts $PbCl_2$, F.P. = 764° C.
20 " KCl + 5 " $PbCl_2$, F.P. = 714° C.
20 " KCl + 10 " $PbCl_2$, F.P. = 633° C.

Less volatilization than the mixture of NaCl + $PbCl_2$.

Mixtures of Calcium Chloride and Lead Chloride. —

20 parts $CaCl_2$ + 2 parts $PbCl_2$, F.P. = 764° C.
20 " $CaCl_2$ + 5 " $PbCl_2$, F.P. = 746° C.
20 " $CaCl_2$ + 10 " $PbCl_2$, F.P. = 710° C.

Mixtures of Sodium Chloride and Potassium Chloride. —

20 parts NaCl + 2 parts KCl, F.P. = 787° C.
20 " NaCl + 5 " KCl, F.P. = 750° C.
20 " NaCl + 10 " KCl, F.P. = 686° C.

Mixtures of Sodium Chloride and Calcium Chloride. —

20 parts NaCl + 2 parts $CaCl_2$, F.P. = 790° C.
20 " NaCl + 5 " $CaCl_2$, F.P. = 722° C.
20 " NaCl + 10 " $CaCl_2$, F.P. = 720° C.

Artificial Lead Sulfide. — F.P. = 1020°–1030° C.; melted in magnesia crucible.

Galena. — F.P. = 950°–1010° C.

Different ores were taken, the M.P.'s varying between 950° C. and 1010° C.

The artificial lead sulfide was prepared by mixing 8 pounds litharge, 2¾ pounds powdered sulfur, and 128 grams argols, and melting under a covering of a mixture of borax glass and soda. The temperature of the furnace was raised slowly up to white heat, and held there a few minutes, then the crucible was removed from furnace and allowed to cool.

Sp. gr. of artificial galena = 7.64.

It contained 85.64 per cent. lead. (Theoretical for PbS = 86.61 per cent. Pb.)

Reduction of Lead Sulfide by Electrolysis.

Having determined the freezing point of a series of salts, and mixtures of salts, the ones most suitable for electrolytes were selected and used in the following experiments:

The method of reducing the lead sulfide was to crush it to about 30 mesh, and float it on the surface of a fused metallic lead cathode, in a fused salt electrolyte. The anodes were strips of Acheson graphite. The cells were either clay or graphite crucibles, 3½ inches in diameter, and heated in a small gas-blast furnace. Clay crucibles were used for chlorides, and graphites for fluorides. The temperature of the furnace was maintained at such a point that the electrolyte was in a thoroughly fused condition. Upon passing

the current, electrolytic decomposition resulted; at the start, when chloride electrolytes were used, chlorine was liberated, after a few minutes sulfur chlorides were given off, and finally sulfur, which ignited at the surface of the bath, forming sulfur dioxide.

A Le Chatelier thermo-electric couple, protected by a suitable tube, was placed in the fused bath at intervals in order to record the temperature at which the electrolysis was conducted.

From time to time, lead sulfide was added to the cell to replace that which was being reduced; the time when more lead sulfide was needed being indicated by the liberation of sulfur chlorides, which even in smallest amounts have a characteristic odor.

After the completion of each run, the contents of the crucible was emptied into an iron mould, and the anode and the crucible scraped clean. After the mass had cooled, the metal was broken from the salt, and weighed. The loss in weight of the anode was also noted. From this date, the loss of materials and the current efficiency was calculated. The latter was obtained by multiplying the ampere-hours of current which passed through the cell by 3.86 (1 A. H. reduces 3.858 grams lead); then dividing the weight of the recovered lead by this value.

The first series of runs were made with artificially prepared galena, and after finding the conditions which were best suited for its reduction, natural galena ore was used.

In order to protect the Acheson graphite crucibles from the rapid corrosive action of the blast flame, they were covered on the outside with a thin coating of an emulsion of water glass and powdered carborundum.

Experiment 1.

Cell: Graphite crucible 3 inches diameter × 2½ inches deep, inside.

Anode: Acheson graphite ½ inch diameter × 12 inches long.

Electrolyte: Fused mixture of 180 grams cryolite, 90 grams sodium chloride and 30 grams potassium chloride.

Weight of lead sulfide added = 256 grams.

Temperature of electrolyte during run 770°–810° C.

Time of run = 3½ hours.

Amperes passing 25–35; average 32.2 amperes.

E.M.F. between anode and cell = 6 to 12 volts, average = 9.5 volts.

C.D. at anode = average 2,040 amperes square foot.

C.D. at cathode = average 384 amperes square foot.

$32.2 \times 3\frac{1}{2} = 112.7$ ampere hours.

$112.7 \times 3.86 = 435$ grams lead, theoretically.

Lead reduced was 205 grams.

Efficiency of current = 47 per cent.

Considerable salt lost by volatilization, sublimed and collected on sides of furnace and on cooler parts anode.

Experiment 2.

Cell: Graphite crucible 3 inches diameter × $2\frac{1}{2}$ inches deep, inside.

Anode: Acheson graphite $\frac{1}{2}$ inch diameter × 12 inches long.

Electrolyte: Fused mixture 280 parts NaCl, 70 parts KCl, and 72 parts $CaCl_2$.

Weight of lead sulfide added in small quantities at time was 500 grams.

Added 80 grams of metallic lead to serve as a nucleus for the reduced lead.

Temperature of electrolyte during run 750°–880° C.

Time of run 4 hours.

Amperes passing = 16–35. Average = 27.6 amperes.

E.M.F. between anode and cell = 5 to 14 volts. Average = 9.2 volts.

C.D. at anode = average 1,944 amperes square foot.

C.D. at cathode = average 331 amperes square foot.

$27.6 \times 4 = 110.4$ ampere hours.

$110.4 \times 3.86 = 426$ grams lead, theoretically.

Weight of lead = 242.

Lead reduced = 242 — 80 = 162 grams.

Efficiency of current = 38 per cent.

Chlorine was given off from time to time in large quantities.

The evolution of sulfur was intermittent, and accompanied by yellow dancing flames.

Rocking the anode assisted the operation, as did the heaping of the lead sulfide around the side of the cell, keeping the center free.

Towards the end of the run, 20 grams of borax glass was added, which somewhat prevented the volatilization of the fused salts, which continued in large quantities until addition of the borax. The sublimed salts collected on the furnace and on colder parts of the anode.

Experiment 3.

Cell: Graphite crucible 3 inches diameter × 2½ inches deep, inside.

Anode: Acheson graphite ½ inch diameter.

Electrolyte: Fused mixture of 185 grams cryolite and 65 grams fluorspar.

Weight of lead sulfide added, small amounts from time to time = 500 grams.

Temperature of electrolyte during run = 950°–1075° C.

Time of run = 3 hours.

Amperes passing = 27–33; average = 30.5 amperes.

E.M.F. between anode and cell = 8–13 volts. Average 9.0 volts.

C.D. at anode = average 2,160 amperes square foot.

C.D. at cathode = average 366 amperes square foot.

30.5 × 3 = 91.5 ampere hours.

91.5 × 3.86 = 353 grams lead, theoretically.

Lead reduced was 275 grams.

Efficiency of current = 77.9 per cent.

During the run sulfur fumes were freely given off.

The bath at the start was quite fluid, but gradually grew more viscous, until finally it became so sticky that the lead did not collect well, so electrolysis was discontinued. The crucible was heated to high temperature in order to allow the particles of lead to settle in one mass, it was allowed to cool and the lead button broken from the salts.

Experiment 4.

Cell: Sand crucible 4 inches diameter × 4¼ inches deep.

Anode: Acheson graphite ½ inch diameter.

Electrolyte: Fused calcium chloride.

Added 100 grams of metallic lead as a nucleus for the collection of the reduced metal. It also served as the cathode, being brought into electrical contact with the source of current by means of an iron rod protected by a porcelain tube.

Weight of lead sulfide, added in small quantities from time to time, was 677 grams.

Temperature of electrolyte during run = 810°–1025° C.

Time of run = 5½ hours.

Amperes passing = 30–45; average 39 amperes.

E.M.F. between anode and cathode = 18–22 volts; average 20.4 volts.

C.D. at anode = average 2,808 amperes square foot.

C.D. at cathode = average 546 amperes square foot.

39 × 5½ = 214.5 ampere hours.

214.5 × 3.86 = 828 grams lead, theoretically.

Lead reduced was 560 — 100 = 460 grams.

Efficiency of current = 55.67 per cent.

Before emptying the contents of the crucible into an iron mould, the temperature of the furnace was raised to about 1200° C., so as to give any particles of suspended metal a chance to collect with the main bulk of lead.

Experiment 5.

Cell: Sand crucible 4 inches diameter × 4¼ inches deep.

Anode: Acheson graphite ½ inch diameter.

Electrolyte: Fused calcium chloride.

Added 370 grams of metallic lead to serve as the cathode and as collector. Diameter exposed = 4 inches.

Weight of lead sulfide added, from time to time, 723 grams.

Temperature of electrolyte during run = 775°–980° C.

Time of run 4¼ hours.

Amperes passing = 38–46; average 42.7 amperes.

E.M.F. between electrodes = 12–22 volts; average = 14.6 volts.

C.D. at anode = average 2,135 amperes square foot.

C.D. at cathode = average 491 amperes square foot.

42.7 × 4¼ = 180.5 ampere hours.

180.5 × 3.86 = 697 grams lead, theoretically.

Lead reduced was 730 — 370 = 360 grams.

Efficiency of current = 51.7 per cent.

The loss in weight of anode (consumption) = 3.5 grams.

Raised temperature of furnace to about 1100° C., then poured metal into an iron mould.

Considerable sulfide of lead remaining.

Experiment 6.

Cell: Sand crucible 4 inches diameter × 4½ inches deep.

Anode: Acheson graphite square 1 inch × 1 inch.

Electrolyte: Fused calcium chloride.

Added 407 grams of metallic lead to serve as the cathode and as the collector. Diameter exposed = 4 inches.

Weight of lead sulfide added, from time to time, 800 grams.
Temperature of electrolyte during run = 848°–908° C.
Time of run = 4½ hours.
Amperes passing = 38–50; average = 44.8 amperes.
E.M.F. between electrodes = 18–24 volts; average = 22 volts.
C.D. at anode = average 1,344 amperes square foot.
C.D. at cathode = average 515 amperes square foot.
44.8 × 4½ = 201.6 ampere hours.
201.6 × 3.86 = 778 grams lead, theoretically.
Lead reduced was 770 — 407 = 363 grams.
Efficiency of current = 46.67 per cent.
The loss in weight of anode (consumption) = 3.3 grams.

Experiment 7.

Cell: Sand crucible 4 inches diameter × 4¼ inches deep.
Anode: Acheson graphite, square 1 inch × 1 inch.
Electrolyte: Fused calcium chloride.

Added 423 grams metallic lead to serve as the cathode, and as the collector. Diameter exposed = 4 inches.

Weight of lead sulfide added, from time to time, 600 grams.
Temperature of electrolyte during run 875°–990° C.
Time of run = 4 hours.
Amperes passing = 37–49; average = 39.4 amperes.
E.M.F. between electrodes = 18–28 volts; average = 24.6 volts.
C.D. at anode = average 1,143 amperes square foot.
C.D. at cathode = average 453 amperes square foot.
39.4 × 4 = 157.6 ampere hours.
157.6 × 3.86 = 608 grams lead, theoretically.
Lead reduced was 848 — 423 = 425 grams.
Efficiency of current = 70 per cent.
Loss in weight of anode (consumption) = 6.5 grams.

After stopping the current, the temperature of the furnace was raised to about 1200° C., so as to allow small particles of suspended lead to collect with the large mass. On cooling the lead button was broken from the salts.

Experiment 8.

Cell: Sand crucible 4 inches diameter × 4¼ inches deep.
Anode: Acheson graphite, square 1 inch × 1 inch.
Electrolyte: Fused calcium chloride.

Added 335 grams of metallic lead to serve as cathode and as nucleus for collecting the reduced metal.

For this experiment used 610 grams of natural lead sulfide ore, containing 66.9 per cent. Pb.

Temperature of electrolyte during run = 837°–923° C.

Time of run = 4½ hours.

Amperes passing = 35–43 ; average = 38.5 amperes.

E.M.F. between electrodes = 19–40 volts ; average = 32.7 volts.

C.D. at anode = average 1,116 amperes square foot.

C.D. at cathode = average 441 amperes square foot.

38.5 × 4½ = 173.2 ampere hours.

173.2 × 3.86 = 668.5 grams lead, theoretically.

Lead reduced was 512 — 335 = 177 grams.

Efficiency of current = 26.5 per cent.

Loss in weight of anode (consumption) = 14 grams.

As the efficiency of this run was so low it was suspected that the liberated calcium instead of reducing the galena had alloyed with the lead. Tests were made, and it was found to be the case. The lump of lead which weighed 512 grams, and the fused mass of calcium chloride, weighing 500 grams, were replaced in the crucible and remelted. To the fused mass was added natural galena ore, containing 66.9 per cent. Pb, ground to pass 30 mesh. The temperature of the mass was raised to about 1200° C. and held there for about fifteen minutes, then allowed to cool, and the metal broken from the salts. The weight of the metallic lead was increased to 622 grams ; the amount of lead reduced by this operation being 110 grams. This increased the current efficiency to 43 per cent.

One of the tests to determine that the lead cathode was alloyed with calcium was heating it to about 1200° C. when a brilliant crimson flame characteristic of calcium appeared over the surface of the fused mass.

SUMMARY OF FREEZING POINTS OF SALTS.

$NaCl$ = 792° C.	NaF = 965° C.
$CaCl_2$ = 772° C.	PbF_2 = 552° C.
KCl = 769° C.	KF = 262° C.
$PbCl_2$ = 494° C.	PbS = 1025° C.
$MgCl_2$ = 158° C.	Galena = 950°–1010° C.

Fluorspar (CaF_2) infusible at 1200° C.

Cryolite ($AlF_3 \cdot 3NaF$) 935°–990° C.

Mixtures.

10	parts	NaCl	+ 1	part	KCl	= 787° C.
5	"	NaCl	+ 1	"	KCl	= 750° C.
2	"	NaCl	+ 1	"	KCl	= 686° C.
10	"	NaCl	+ 1	"	$CaCl_2$	= 790° C.
5	"	NaCl	+ 1	"	$CaCl_2$	= 722° C.
2	"	NaCl	+ 1	"	$CaCl_2$	= 720° C.
10	"	NaCl	+ 1	"	$PbCl_2$	= 770° C.
5	"	NaCl	+ 1	"	$PbCl_2$	= 761° C.
2	"	NaCl	+ 1	"	$PbCl_2$	= 723° C.
10	"	KCl	+ 1	"	$PbCl_2$	= 764° C.
5	"	KCl	+ 1	"	$PbCl_2$	= 714° C.
2	"	KCl	+ 1	"	$PbCl_2$	= 633° C.
10	"	$CaCl_2$	+ 1	"	$PbCl_2$	= 764° C.
5	"	$CaCl_2$	+ 1	"	$PbCl_2$	= 746° C.
2	"	$CaCl_2$	+ 1	"	$PbCl_2$	= 710° C.
10	"	Cryolite	+ 1	"	NaCl	= 955° C.
5	"	"	+ 1	"	NaCl	= 905° C.
2	"	"	+ 1	"	NaCl	= 862° C.
10	"	"	+ 1	"	KCl	= 940° C.
5	"	"	+ 1	"	KCl	= 909° C.
2	"	"	+ 1	"	KCl	= 900° C.
10	"	"	+ 1	"	NaF	= 956° C.
5	"	"	+ 1	"	NaF	= 910° C.
2	"	"	+ 1	"	NaF	= 877° C.
10	"	"	+ 1	"	KF	= 955° C.
5	"	"	+ 1	"	KF	= 920° C.
2	"	"	+ 1	"	KF	= 903° C.
10	"	"	+ 1	"	CaF_2	= 960° C.
5	"	"	+ 1	"	CaF_2	= 895° C.
2	"	"	+ 1	"	CaF_2	= 910° C.

Summary of Experiments.

Experiment No.	Electrolyte	Average Amperes.	C.D. Per Sq. Ft. at Anode.	C.D. Per Sq. Ft. at Cathode.	Average Voltage.	Temperature of Electrolyte.	Time of Run.	Lead Reduced.	Current Efficiency.	Loss in Weight of Anode Per Ampere Hour.
1	6 pts. cryolite, 3 " NaCl, 1 pt. KCl	32.2	2040	384	9.5	770°–810 °C.	3½ hrs.	205 grs.	47.0 %	0.032 gram.
2	4 pts. NaCl, 1 pt. KCl, 1 " $CaCl_2$	27.6	1944	331	9.2	750 –880 C.	4 hrs.	162 grs.	38.0	0.014 gram.
3	3 pts. cryolite, 1 pt. CaF_2	30.5	2160	366	9.0	950 –1075 C.	3 hrs.	275 grs.	77.9	0.045 gram.
4	$CaCl_2$	39.0	2808	546	20.4	810 –1025 C.	5½ hrs.	460 grs.	55.6	0.021 gram.
5	$CaCl_2$	42.7	2135	491	14.6	775 –980 C.	4¼ hrs.	360 grs.	51.7	0.019 gram.
6	$CaCl_2$	44.8	1344	515	22.0	848 –908 C.	4½ hrs.	363 grs.	46.6	0.016 gram.
7	$CaCl_2$	39.4	1143	453	24.6	875 –990 C.	4 hrs.	425 grs.	70.0	0.041 gram.
8*	$CaCl_2$	38.5	1116	441	32.7	837 –923 C.	4½ hrs.	287 grs.	43.0	0.080 gram.

*The material treated was natural galena ore, containing 66.9 % lead.

Conclusions.

1. Fused alkali chlorides or fused calcium chloride serve as electrolytes for the cathodic reduction of galena. The former have the advantage over calcium chloride in that the voltage of decomposition is much less. An E.M.F. of about 5 volts is required for the decomposition of the alkali chlorides and sulfides, whereas about 20 volts is required for calcium chloride and sulfide.

2. Fused calcium chloride has an advantage over fused alkali chlorides for the cathodic reduction of galena, in that it is less readily volatilized, and is also a more fluid bath.

3. At temperatures above 1300° C., the fused alkali and calcium chlorides rapidly attack sand and clay crucibles, forming a hard glassy slag. Below 1000° C., the crucibles are scarcely affected other than becoming impregnated with the fused salts.

4. The higher current efficiency, in the case when a fused electrolyte of cryolite and fluorspar was used, is accounted for by the fact that aluminium does not readily alloy with lead, for which reason it floats on the lead and is in more intimate contact with the galena, than is the case with sodium or calcium, both of which readily alloy with lead.

5. A great disadvantage of the use of fluoride electrolytes is that the bath becomes quite viscous with use, which is due no doubt to the solubility of lead sulfide in the molten mass.

6. The chlorides have quite an advantage over the fluorides in that when the bath becomes contaminated with foreign matter, such as SiO_2, CaO, FeO, etc., the chloride salts may be dissolved by water, recrystallized, and are ready to be reused.

7. At the beginning of the operation of electrolysis, for the cathodic reduction of galena, when chloride electrolytes are used, considerable chlorine is given off, the calcium or sodium which is at the same time freed reduces the galena forming metallic lead and calcium or sodium sulfides. After a certain amount of calcium or sodium sulfide has formed, the decomposition of the chloride ceases, because the E.M.F. of decomposition for sulfides is less than it is for chlorides.

8. The sulfur, which is liberated at the anode, distills and may be collected in its elemental form, or allowed to oxidize and escape as SO_2 fumes.

9. The highest current efficiencies, when chloride electrolytes were used for the cathodic reduction of galena, were when the

temperature of the electrolyte was highest. The high temperature is necessary to bring about reaction between the galena and the alloy of lead with either sodium or calcium.

10. The very low current efficiency of the run, when natural galena ore was treated, was due to it being a poorer electrical conductor than the artificially prepared lead sulfide, and also the temperature of the electrolyte did not exceed 923° C. However, when the mass was remelted, under a covering of fused salts, and held at a temperature of about 1200° C., the calcium, which had alloyed with the fused lead cathode, was liberated and reduced the galena which was floated on the lead.

11. No lead is lost by volatilization during cathodic reduction of galena, in a fused electrolyte, as the reduced lead is covered with a bath of fused salts, which is below the boiling point of the metal.

12. Fused electrolytes, containing lead salts, even though having lower freezing points are not suitable for the cathodic reduction of galena, because as lead stands lower in the E.M.F. series than the alkalies and alkaline earths, it would be the first constituent to decompose, thus robbing the electrolyte of the constituent which lowered its freezing point.

METALLURGICAL ABSTRACTS.

By WILLIAM CAMPBELL.

Aluminium.

The Metallurgy of Aluminium in 1906: J. W. Richards. *E. and M. J.*, 83, 1083. Deals with the Aluminum Co. of America, the British Aluminium Co., the Aluminium Industrie aktien Gesellschaft, the Société Electrométallurgique Française, the Société des Produits Chimiques d'Alais et Carmague. See *Mineral Industry*, XV., 15.

Progress in the Use of Aluminium in 1906: J. W. Richards. *E. ana M. J.*, 83, 1147. Deals with aluminium casting, plating, soldering, welding, etc.

Antimony.

Antimony: J. T. Glidden. *Min. Ind.*, XV., 32. The production of refined antimony in the U. S. confined to the Mathison smelter at Chelsea, Staten Island, the Chapman Smelter, San Francisco, and a new plant opened by C. Soloman at Oakland. Sb ores must contain 50 per cent. Sb to be considered first-class ore. Pure sulphide runs 71.8 per cent. Sb. Mechanical dressing of low-grade ore is difficult. Hillgrove, N. S. W., produced 2,451 tons in 1906. The Dominion Antimony Co., Nova Scotia, produced 562 long tons assaying 40 per cent. Sb and 1.5 oz. Au. China exports about 2,000 tons each of ore and regulus. For metallurgy of Antimony, see Vols. I., II., X. Alloys, Siemens-Halske patent bearing metal runs 45 Zn, 45 Cd, 10 Sb.

Bismuth.

Bismuth: E. and M. J., 83, 23. The chief bismuth smelters and refiners are Johnson Matthey of London, the royal smelters of Freiberg and Oberschlema, Saxony, and the Deutsche Gold- und Silberscheideanstalt of Frankfurt. These three form the trust which determines the market price. The prices of ores are given. See: *Bismuth:* W. R. Ingalls. *Min. Ind.*, XV., 81. Also Vols. II., V., VIII.

Copper.

Copper: W. R. Ingalls. *Min. Ind.*, 1907, 191. Gives the production; refining capacity of the 9 important refineries:

Works.	Location.	Capacity, Pounds.
Nichols Copper Co.,	Laurel Hill, N. Y.,	288,000,000
Raritan Works,	Perth Amboy, N. J.,	288,000,000
A. S. and R Co.,	Perth Amboy, N. J.,	144,000,000
U. S. Metals R. Co.,	Chrome, N. J.,	144,000,000
Baltimore,	Baltimore, Md.,	130,000,000
Balbach S. & R. Co.,	Newark, N. J.,	75,000,000
Boston and Montana,	Great Falls, Mont.,	30,000,000
Tacoma S. Co.,	Tacoma, Wash.,	18,000,000
Mountain Copper Co.,	Oakland, Cal.,	3,000,000

Takes up copper mining by states, then copper in foreign countries, etc.

Copper: Progress in the Metallurgy of: L. S. Austin, *Min. Ind.*, XV., 245. A review of the best articles published during the year.

Copper Smelting in Modern Times: The chief Advances in: W. Gowland. *Bull. 31, I. M. and Met.* Traces the development of the blast furnace, with three old Japanese illustrations. Then takes up pyritic smelting, which was first used by John Hollway. The history and development of the reverberatory smelter is given, also that of the converter and refining furnaces,

The Cananea Copper Deposits: R. B. Brinsmade. *M. and Mins.*, XXVII., 465. Deals with concentration plant, smelter capacity 2,000 to 2,500 tons daily. 8 blast furnaces, 6 converter stands, a 220 ton reverberatory and McDougall roaster. See also: *Smelter Charge Handling in the South West: M. and Mins.*, XXVII. (Jan., 1907).

The Copper of Shasta County, California: D. F. Campbell. *M. and S. P.*, XCIV., 55. Gives views of the Keswick Smelter, the Kennett Smelter and United States Smelter.

The Washoe Smelter: M. and S. P., XCIV., 467. Sloping site. Concentrator plant in 4 units each handling 1,100 tons ore per 24 hrs. 64 Evans-Klepetko-McDougal roasters. 8 reverberatories 19 × 102 to 116 ft. 300 tons per 24 hrs. Briquette plant with 4 Chambers Bros. end cut auger-type machines 700 tons each per 24 hrs. 3 B. F. two 51 ft. and one 87 ft. × 56 in. take 1,600 and 3,000 tons respectively.

The Boundary District, British Columbia: Forbes Rickard. *M. and S. P.*, XCIV., 511. Gives view of the Granby Smelter, which works on ore $1.25 gold, 27 to 30 lbs. copper.

The Wallaroo Smelting Works: T. C. Cloud. *Inst. M. and Met.*, Nov. 15, 1906; *E. and M. J.*, 83, 324. The plant is situated on the west coast of Yorke's peninsula, South Australia and consists of copper sulphate works, capacity of 500 to 600 tons per an.; sulphuric acid works with output of 5,000 tons per an.; smelter for refractory gold ores using copper as a collector, an electrolytic department turning out 12 tons of copper per week; an electrolytic plant for parting gold and silver. Details of the work are given with an excellent plan of the smelter.

The Copper-Smelting Works at Humboldt, Ariz.: E. H. Hamilton. *E. and M. J.*, 83, 901. Concentrates roasted in Edwards furnaces, smelted in reverberatories and bessemerized. Coarse ore smelted in blast furnace.

United States Smelting, Refining and Mining Co.: E. and M. J., 83, 993. Gives a list of the subsidiary companies of the corporation. During the year the directors authorized an enlargement of the Mammoth Smelter, Calif., improvements at the Bingham Junction Smelter, a lead refinery at Grasselli and an enlargement of the Chrome plant.

Chrome Plant of U. S. Metals Refining Co.: L. Addicks. *E. and M. J.*, 83, 1001. The refinery has been brought up to a capacity of 200 tons a day and a complete copper smelting plant added. The article describes ore sampling, two blast furnaces, two converter stands, three anode furnaces, tank house of 816 tanks, three wirebar furnaces,

power plant, parting plant where the doré is parted electrolytically in cells of the Thum type.

Pyritic Smelting: The Development of: R. C. Sticht. *Trans. Aust. Inst. M. E.*, XI., p. 1.

Copper-Converter: A Modern Electrically-Operated: G. B. Shipley. *Elect. and Met. Ind.*, V., 244. Describes new plant of three stands of the Oxford Copper Co. at Constable Hook, N. J.

Bessemerizing Copper Mattes: Relative Elimination of Iron, Sulphur and Arsenic in: E. P. Mathewson. *Bull. A. I. M. E.*, 1907, 7. Gives results of experiments at the Washoe works. Charts show percentage composition for Cu, S, Fe and As to time of blowing in minutes.

Copper Converters, Lining and Drying: H. L. Charles. *E. and M. J.*, 83, 1046. Replaces the fuel-drying method by the use of molten converter slag or low grade matte.

Slag from Smelting Works: Disposal of: H. Lang. *E. and M. J.*, 83, 1240. The breakwater at Santa Rosalia, Lower California is being constructed from the slag of the Compagnie du Boleo Copper Smelter.

Slag-Casting Machine: L. S. Austin. *M. and S. P.*, XCIV., 282. Shows the Bennetts Slag-Casting Machine installed at Tacoma.

The McMurtry-Rogers Process for Desulpherizing Copper Ores and Matte: T. C. Cloud. *Bull. 31, I. M. and Met.* Describes the use of the H. and H. converter for ore at the Wallaroo Works, charge runs 15–35 per cent. SiO_2 and 15–25 per cent. S., weighs 8 tons net. False bottom covered with pieces of previously burnt charge 3–4 inches in size. Fire started on top and pot one half filled with ore, when this has become thoroughly ignited the pot is filled.

Gold and Silver.

Gold and Silver: F. Hobart. *Min. Ind.*, XV., 340. Deals with production in U. S. by states, in foreign countries, commercial features, etc.

Gold Milling in 1906: Progress in: R. H. Richards and C. E. Locke. *Min. Ind.*, XV., 391. Review of literature on stamp mill and stamp mill practice. Tube mills, concentration of black sands (Day and Richards, Mineral Resources for 1905).

Gold Ore Treatment during 1906: Progress in: Alfred James. *E. and M. J.*, 83, 17. Deals with crushing, fine grinding, amalgamation, roasting, concentration, treatment of difficult ores, slimes, treatment, tailings, re-treatment, mercuric cyanide and costs. See also *M. I.*, XV., 402.

Gold Dredging in 1906: J. P. Hutchins. *E. and M. J.*, 83, 21. Discusses the testing of dredging ground, expansion of dredging, procedure, design and cost. See also *M. I.*, XV., 425.

The Great Gold Mines: T. A. Rickard. *M. and S. P.*, XCIV., 565. Are as follows: Simmer and Jack, Robinson, Robinson Deep, Cason and Village Main Reef (Transvaal), Esperanza (Mex.), Homestake, Camp Bird, Champion Reef, Mysore (India), Waihi (N. Z.), Mt. Morgan (Queensland), Oroya-Brownhill, Great Boulder, Golden Horseshoe, Ivanhoe (W. Australia).

Gold Milling: Modern: G. P. Scholl. *Electro. and Met. Ind.*, V., 14. General review embracing stamps, tubemills, filters, etc. Liberty Bell Mill Telluride, Colo.: the Smuggler Union: Standard Mill, Bodie, Calif.; are mentioned.

The Gold Prince Mine and Mill: Scholl and Herrick. *M. and Mins.*, XXVII., 337. The mill is at Animas Forks, Colo. Cost $500,000. 100 stamp battery, 12 ft. x 4½ ft. plates. 76 concentrating tables of Card pattern. 5 tube mills. The paper includes a flow sheet and several illustrations.

Milling at Gladstone, Colo.: G. P. Scholl. *M. and Mins.*, XXVII., 498. The Gold King mill has 80 stamps, plates, 16 Card tables, 4 tube mills 20 ft. x 5 ft. Allis Chalmers type, from which the pulp goes to 4 Card tables. The Mogul Mill has 40 stamps, 15 Card tables. Slimes treated on Card and Wilfley tables. Four products, lead, zinc iron, silica and tailings. Blake-Mosher separators.

Tube Mills at the Waihi Gold-Mine, New Zealand: Grinding in: E. G. Banks. *Bull. A. I. M. E.*, 1907, 63. The tube mills increase extraction 36 c. per ton of ore. Increase the tonnage, save 75 per cent. of screen costs and improve amalgamation.

Tube Mills — The Economics of: Mines and Minerals, XXVII., 297. Abs. from South African Mines, October 27, 1906. There are 58 tube mills on the Rand. The initial outlay of stamps and tube mills is contrasted and the latter are the cheaper.

Cyanidation in the United States and Mexico: C. H. Fulton. *Min. Ind.*, XV., 410. Reviews slime treatment, Butters Filter, Moore Process at the Liberty Bell, the Merrill Filter Press. Cyanidation of silver ores. New Milling Practice.

Cyanidation During 1906: Progress of: A. James. *M. and S. P.*, XCIV., 18. Stage crushing is in great evidence. 1,500 lb. stamps are used on the Rand. For fine-grinding tube mills are generally employed. At Kalgoorlie pans are used. Where cyanide solutions circulate through the mill the copper plates are scoured, and in places amalgamation is now done in pans. Some figures in roasting are given. Under the heading of slime treatment the Ridgway filter press is described and other methods mentioned. The review ends with a table of costs for the Lake View, Ivanhoe and South Kalgurli.

Cyanide Process: Recent Improvements in: F. L. Bosqui. *M. and Mins.*, XXVII., 298. Deals with the early methods, and history of the process. Tube-milling and filter-pressing are the two great improvements. Slime filters are discussed.

The Butters Slime-Filter at the Cyanide Plant of the Combination Mines Co., Goldfield, Nev.: M. R. Lamb. *Bull. A. I. M. E.*, 1907, 67. The slime averages $20 per ton. Drawings of a 20 frame filter of 40 tons capacity are given and method of work described.

The Butters Filter: Mark R. Lamb. *M. and S. P.*, XCIV., 152. A drawing and photograph show the filter and mode of operation.

The Ridgway Filter: M. and S. P., XCIV., 181. Machine built at Great Boulder Mine, Kalgoorlie. Circular 12 ft. dia. 12 flat cast-iron filtering frames in form of sectors of a circle. The trough is divided into three compartments, one for slime-pulp, one for wash water,

and one for residue discharge. The track has elevations so that the filtering frames are lifted from one compartment to the next. Working cost 3.7 pence per ton.

Butters Method, Filtration of Slime by: E. M. Hamilton. *M. and S. P.*, XCIV., 785, 818. Tailings from Comstock mills contain 75 per cent. of unleachable material or slime. Settles best in distilled water. Butters vacuum filter is described, with drawings. Mode of procedure is given together with working costs at Virginia City. Total working cost of 8 c. per ton of slime.

Butters Cyanide Plant, Virginia City, Nev.: C. T. Rice. *E. and M. J.*, 83, 269. Built to treat old pan-amalgamation tailings; also treats some siliceous ores. The pan tailings contain considerable copper. They are treated direct, and the resulting solution goes to the Siemens-Halske boxes where the solution is regenerated. The tailings are pulped in cyanide solution, separated into slimes and sands, the slimes treated by agitation and filtration, the sands percolated. A description of the treatment of ore follows, the Blaisdell equipment, the Butters filter plant, Siemens-Halske electric precipitation, etc. The article is well illustrated by photos and drawings.

The Homestake Slime Plant: M. Ehle. *M. and Mins.*, XXVII., 358. The plant will treat by cyanidation by pressure lixiviation in filter presses the slimes resulting from the daily crushing by stamps of some 4,000 tons of ore. The pulp is separated by water classification into some 2,300 tons of sands and 1,700 tons of unleachable slimes which have heretofore gone to waste. A flow sheet of scheme of separation into sands and slimes is given and the process described with several photographs.

On the Rand: M. and S. P., XCIV., 632. Shows a 400-stamp mill in construction (1898) for the Simmer East and Knights Deep mines. Mill with tube mills now crushes nearly 8 tons per stamp. Mill with cyanide and slime plant, sorting house, precipitation house and refineries cost about $2,500,000. A 60 ft. tailings wheel is also shown.

Langlaagte Deep, Ltd, Visit to: Trans. Transvaal Inst. Mech. Engs., Feb., 1907, p. 191. Abs. *J. C. Met. and M. Soc. S. A.*, VII. 348. Gives the equipment of the Blaisdell sands handling plant. One class A. Blaisdell excavator, Robins conveyor belts; class ZX Blaisdell distributor. Excavator does 88 tons per hr.

Metallurgical Chart of Operations in the Mill of the Butters Copala Mines in Sinaloa, Mexico: L. N. Bullock. *M. and S. P.*, XCIV., 221. Shows batteries, tube mills, cones for sand and slime, 3 collecting vats, 10 sand leaching vats, 6 slime treatment vats, Butters vacuum filter, zinc boxes. See also p. 335.

Guanajuato, Mexico: Old and New Methods at: T. A. Rickard. *M. and S P.*, XCIV., 824. Mill of the Guanajuato Reduction and Mines Co. 80 stamps 30-mesh, concentrate on Wilfleys, middlings reground in Abbé tube mill. Tails to thickening cones. 8 in. cast-iron pipe to cone classifiers in the Hacienda Flores. Sands distributed by Butters and Mein device to two 350-ton receiving vats. Discharge to belts. 8 leaching vats. 14-day treatment with 0.5 per cent. KCN. Wash

and discharge by belts into river. Slime treated by mechanical stirring and large centrifugal pumps. 4 days. Settling vats. Slime discharged with a very small percentage of moisture into river. Costs of patio process are given. Total 12.56 Pesos.

Guanajuata, Metallurgical Development at: T. A. Rickard. *M. and S. P.*, XCIV., 627. The Sirena mill succeeded an old patio, with 20 stamps, Boss pans and settlers, 5 Wilfleys. Extracted 77 per cent. As the ore went down in depth the extraction fell off — concentration preceded amalgamation. Again extraction fell. Cyanide plant put up in 1905. Flow from Wilfleys goes to cone classifiers and slimes and sands treated separately. Sands run 297.5 gm. Ag, 1.37 gm. Au; after treatment 52 gm. Ag and 0.1 gm. Au. The slimes run 275.5 gm. Ag and 1.3 gm. Au; after treatment 45.5 gm. Ag and 0.1 gm. Au.

Cyanide Plant for Treating Guanajuata Ores: C. W. Van Law *E. and M. J.*, 83, 649. 80 stamp mill treats over 7,500 tons a month. The cyanide plant uses double-cone system to classify into sands and slimes; Butters distributors, etc., for sands; slimes agitated by 6-in. Meese-Gottfried centrifugal pumps supplemented by mechanical arms. Decantation. Zinc precipitation.

Blaisdell Apparatus at El Oro Cyanide Plant: E. and M. J., 83, 230. The tailings from the mill launder go to wet-pulp distributor for nine 22 × 10 ft. steel collecting vats in 3 rows. Slimes overflow to slime treatment vats. The sand after draining is discharged by a 22-ft. excavator, thence by 18-in. belt-conveyors to the leaching vats (two rows of 6 steel vats 40 ft. dia.). After treatment a 40-ft. excavator removes the sand which goes to the stacking conveyors. Plant treats the products of mills Nos. 1 and 2, each of 100 stamps. Four photographs show the work.

Mining Conditions in the Mountains of Chihuahua: J. B. Farish. *E. and M. J.*, 83, 221. The Dolores Mines Co.'s mill has three batteries of five stamps each, weigh 900 lbs., make 102 drops per min. from 6 to 7½ inches. Three Bryan mills for secondary crushing. Four standard 5-ft. pans with two 8-ft. settlers and a clean-up pan. Tube mill of Gates pattern, 16 ft. long, 42 in. in dia. Cyanide plant consists of 4 agitating tanks 22 ft. dia. by 15 ft. deep, three decanting tanks and two final settling tanks 30 ft by 15 ft. In addition there is a sand plant of 8 to 10 tons capacity. The improvements to the mill are mentioned.

The Bullfrog Cyanide Mill: E. R. Ayres. *E. and M. J.*, 83, 376. The new mill will consist of a 15 x 24 in. Blake Crusher, Vezin Sampler and sample crusher. From mill bin ore goes to No. 1 rolls 42 in. dia. by 14 inch face, sized, to No. 2 rolls, screened to No. 3 rolls. Copper plates, amalgam traps, Dorr classifiers, concentrated sands by Wilfleys, slimes by Frue vanners. Tailings cyanided. Sands to receiving tanks with filter bottom and vacuum attachment, then Blaisdell excavator and belts to five leaching tanks 36 x 6½ ft. Four lb. solution used. Zinc precipitation. limes to 2 thickening cones then to 4 slime tanks 28 ft. x 16 ft. deep with 4-arm Blaisdell agitator. Half lb. KCN solution used. Butters filter used.

The Cyanidation of Raw Pyritic Concentrates: F. C. Smith. *Bull. A.*

I. M. E., 1907, 1. Describes a metallurgical campaign at the mines of the Socorro Gold Co., Yuma County, Ariz. Ore was quartz and pyrite, occasional pockets of galena, etc., 20 stamp mill with plates, 3 standard tables, 60-ton cyanide mill at some distance were unsatisfactory. On account of local conditions it was necessary to have a very clean concentration of the mill-tails, producing a final material containing very low values and a local treatment of raw concentrates. Two Frue vanners were installed. It was found that the raw concentrates ground to 100-mesh and agitated yielded a good extraction, 85 per cent. Au and 70 per cent. Ag using 6.5 lb. KCN per ton. 2 tons a day to be treated. A second hand old style amalgamation pan was used and charged with 1 ton sol. 6 lb. KCN, 6 lb. lime, 1.5 ton raw concentrates run at 75 rev. per min. and grinding continued 8½ hrs. An assay showed an extraction of 90.7 per cent. of the values.

Notes on the Occurrence and Treatment of an Auriferous Ore containing Insoluble Arsenides: J. K. Wilson. *J. C. M. and Met. Soc. S. A.*, VII., 244. Gives results of a series of tests on the tailings of the Nooitgedacht Mine, Lyndenburg Dist. After the addition of lime and washing, the tailings are treated with KCN solution of 0.04–0.06 per cent. Time of treatment 8 to 12 days.

Cyanide Practice at the Reliance Mill: D. Lay. *E. and M. J.*, 83, 758. Plant on 49 Creek, Nelson, B. C. Ore a decomposed quartz containing a high percentage of hydrated iron oxide and from 5 to 25 per cent. iron pyrite, $4 in gold. Blake Crusher. 6 ft. Chilean Mill to 60-mesh in alkaline 0.02 per cent. KCy solution. Settling tanks. Electrical precipitation. Ore slimes treated in Hendrix agitator, the solution being filtered during agitation, pumped to the precipitation tanks and returned to the agitator. 12 hours treatment.

Electrolysis of Gold from Cyanide Solutions: D. Lay. *E. and M. J.*, 83, 801. Describes method of electrical precipitation used at Reliance Mill, Nelson, B. C.

Electrolytic Precipitation of Cyanide Solutions: C. P. Richmond. *E. and M. J.*, 83, 512. Describes the mill of the Butters Salvador Mines, Ltd., at the San Sebastian mine near Santa Rosa, Salvador. Complex cupriferous ore. Blake-Marsden crusher, rotary dryer. No. 5 Krupp ball mills to 40-mesh. Roast in Jackling furnaces. Cyanide as sands and slimes. Recovery of gold and copper by combined electrical and zinc precipitation. First of all the Butters Electrical Precipitation process is described and then the details of electrical refining are given. The lead cathodes with their deposit of copper and gold are placed in cotton cloth frames in a solution of 2–3 per cent. sulphuric acid and act as anodes. The cathodes are lead, on which the copper is precipitated, the gold remaining in the cloth bags, thus forming an easy and efficient method of separation.

Cyanide Works Clean-up Practice: J. E. Thomas. *J. Chem. Met. and M. S. S. Africa*, VII., 109. Advocates better acid treatment to dissolve the zinc; the use of filter-papers over the cloths of the press; the use of hot water for washing cakes in the press. Any gold in the wash water may be precipitated again by zinc fume.

The Tavener Process for Gold Slimes: L. A. E. Swinney. *Inst. M.*

and Met., Nov. 15, 1906; *E. and M. J.*, 83, 608. The filter-pressed gold zinc slimes are dried in a reverberatory, assayed, fluxed so that the lead bullion obtained runs 10 per cent. Au and smelted. The bullion is cupelled, remelted in Cornish crucibles and cast. A by-product smelt cleans up all the residues, slags, etc. The total cost is less than 4 d. per fine oz.

Refining of Base Bullion, A few notes on the: C. W. Lee and W. O. Brunton. *J. Chem. Met. and M. Soc. S. A.*, VII., 358. Deals with the refining of gold bullion by means of the injection of air into the molten metal. A pressure cylinder used for the air, ½ in. iron pipe to furnace where one of Morgan's 24 in. by ⅛ in. bore porcelain chlorine tubes is attached, and dips into the metal to be refined. Cyanide bullion 647 fine brought up to 814 in two hours.

Matte from Cyanide Mill, Treatment of: A. E. Drucker. *M. and S. P.*, XCIV., 638. Amount of matte in clean-up depends on completeness of wash after acid treatment and the extent to which the final roast is carried. Matte contains Zu, Pb, Fe, Au and Ag, assays $20 per lb. Matte, borax and KCN are crushed fine, and put in graphite crucible in layers. White heat for 2 to 3 hrs. Skim and pour. Extracts 85 to 94 per cent. of value of original matte.

The Edwards Furnace: M. and S. P., XCIV., 798. Four duplex roasters at Golden Cycle Mill, Colorado City, each handles 100 tons per 24 hrs., to 0.008 per cent. S. 5–6 h.p. Photograph of furnace.

Pan Amalgamation: H. W. Bangle. *M. and S. P.*, XCIV., 826. Gives the theory of the process. Deals with the practical side of the subject, with examples, Los Grados, El Progreso at Pachuca and other mills.

Old Mexican Methods: B. MacDonald. *M. and S. P.*, XCIV., 123. Shows some interesting plates from the report made in 1866 by E. Tillmann, the Royal Commissioner of Mines in Prussia, on Guanajuato.

Assay of Gold Bullion: A. C. Claudet. *Bull. 27. Inst. M. and Met.*, Dec., 1906, p. 15. Describes the preparation of the assay piece and weighing of 1,000 millième (½ grm.) by substitution. Each piece is wrapped in 2 gm. of lead foil with 35 millième of copper wire and two pieces of fine silver each 1,250 millième, and cupelled in a Fletcher gas muffle, in 8 minutes, at a good orange red. The parting is done in platinum tray holding 60 cornets. 1st acid sp. gr. 1.160, 2d acid 1.250 for 20 minutes. 3d acid 1.250 for 20 minutes. Washed and annealed. Fine gold weighed by difference against the 1,000 millième weight. The method of preparing pure gold for proofs is given.

Iron and Steel.

Iron and Steel in 1906. Progress in the Metallurgy of: B. Stoughton. *Min. Ind.*, XV., 480. Deals with ores, briquetting and nodulizing, the bla t furnace, stoves, use of gas engines, cement from slag. Manufacture of steel, open hearth. Mechanical treatment. Founding. Electric smelting and miscellaneous subjects.

Steel Ingots. Piping and Segregation in: H. M. Howe. *Bull. A. I. M. E.*, 1907, 169. Piping is shortened by casting in wide ingots and

in moulds of low conducting power. The pipe is shortened and segregate raised by top-casting, slow casting, with large end up, by retarding cooling of the top, by allowing deep-seated blow holes to form and by liquid compression. Segregation is lessened by quieting steel (Al), probably by casting in small ingots and hastening solidification. The paper is over 100 pages and deals with the subject thoroughly.

The Blast Furnaces at the Minnequa Steel Works: L. Lewis. *E. and M. J.*, 83, 178. Describes the blast furnace plant of the Colorado Fuel and Iron Co., Pueblo. Photographs and a section illustrate the article.

Steel Making at Pueblo: L. Lewis. *E. and M. J.*, 83, 234. Describes the Bessemer plant and the rail mill. Two converters.

Making Open Hearth Steel at Pueblo: L. Lewis. *E. and M. J.*, 83, 371. Describes the Open Hearth plant and its methods, with several excellent drawings and photographs. Six furnaces, six additional 50-ton are building with a 300-ton hot-metal reservoir.

Compound Steel. A Modern Steel Plant Proposition: P. Eyermann. *Electro. and Met. Ind.*, V., 123. Taps directly from B. F. into "compound furnace" where, by a combined open hearth and converter method, steel is produced.

Sampling and Analysing Iron Works Materials: M. Orthey. *Metallurgie*, 1907, 266. Deals with methods and results. Table I. of 60 analyses of pig irons given. Table II. gives comparison of analyses obtained by the buyer, the seller and the umpire.

Hardening Processes: Exhibition of: C. Rosambert. *Rev. de Met.*, 1907, 346. Vienna, May–July, 1906. Gives illustrations of furnaces employed. Steels used and their qualities, etc.

Le Laboratoire du Congrès de l'Association Internationale des Méthodes d'Essais a Bruxelles: L. Guillet. *Rev. de Met.*, 1907, 189. The work consisted of the following tests: (1) Micrographic, (2) determination of critical points, (3) shock tests on nicked bars, (4) Brinell test, (5) Fremont test. The Le Chatelier microscope, the Saladan recorder, the Guillery machine, Brinell machine and Fremont apparatus were used.

LEAD.

Lead Smelting: Recent Improvements in: H. O. Hofman. *Min. Ind.*, XV., 523. Deals with physical properties, alloys, lead ores, smelting rates in British Columbia, works accounts, pot-roasting, smelting in British Columbia, Freiberg (Kochinke: *Jahrbuch für das Berg- und Hüttenwesen im Königreiche Sachsen*, 1902, p. 96, 1906, A., 158), in Austria, at Raibl, at Gailitz, at Scherian, at Kreuth, at Windisch Bleiberg, at Eisenkappel in Carinthia, in Tasmania. Blast furnace and condensation of fume. Lead refining.

The Metallurgy of Lead in 1906: W. R. Ingalls. *E. and M. J.*, 83, 20. Takes up pot-roasting of ores, dust chambers and flues, which in new constructions are made of brick in the form of an inverted catenary. The bag house is coming back into favor. The Betts process is working at Trail, B. C., at Locke, Blackitt and Co., Newcastle on Tyne, and at the plant of the U. S. Metals Refining Co. near Chicago.

Silver and Lead Mines. The Great: T. A. Rickard. *M. and S. P.*, XCIV., 790. Silver: Broken Hill Prop. Co., 1906. produced 4,615,430 oz.; Federal M. and S. Co., 3,920,884 oz.; Tonopah M. Co., 3,194,309 oz.; Anaconda Copper M. Co., 2,979,908 oz.; Bunker Hill and Sullivan M. and Concentrating Co., 1,598,426 oz.; Nipissing Mines Co., 1,482,767 oz.

Lead: Federal Mining (1906), 63,029 tons; Broken Hill Proprietary, 57,248 tons; Bunker Hill and Sullivan, 40,113 tons.

Silver-Lead Smelting Practice: T. S. Austin. *M. and S. P.*, XCIV., 252. Deals with slag composition and characteristics. XCIV., 341, operation of the furnace, quantity of fuel, blast, etc. XCIV., 537, smelting of oxidized ores at Chihuahua. XCIV., 762, Loss of gold in refining base-bullion, matte concentration, treatment of Speiss, etc.

The Adjusting of Ores: L. S. Austin. *E. and M. J.*, 83, 226. Gives chart of schedule of prices of lead ores of the American Smelting and Refining Co. in 1906 for Clear Creek and Gilpin Co. ores.

The Lead-smelting Works of Port Pirie: G. D. Delprat. *E. and M. J.*, 83, 516. Lead concentrates from Broken Hill roasted in Ropp furnaces, mixed with 14 per cent. of fine shell, 4.5 per cent. fine iron ore and 9 per cent. siliceous ore. Each furnace takes 100 tons per day. Seventeen H. and H. pots each hold 8 tons. Thirteen blast-furnaces, 8 in blast each producing 25 to 30 tons bullion per 24 hrs. Charge carries 17 per cent. lead. Slag: 25 per cent. SiO_2: 33 per. cent. FeO: 6 per cent. MnO: 12 per cent. CaO: 13 per cent. ZnO: 6 per cent. Al_2O_3: 3 per cent. S: 1.5 per cent. Pb. The refining and desilverization of the base-bullion is described. Sulphuric acid parting is used.

Tredinnick's Process for Desilverizing Lead: L. S. Austin. *M. and S. P.*, XCIV., 89. An improvement on the Luce and Rozan process. Uses a train of twelve crystallizing kettles 50 tons each and supported by a hydraulic ram by which it can be raised and lowered at will and lead can be run from kettle to kettle as required.

The Faber du Faur Zinc Furnace: E. and M. J., 83, 84. Gives drawings of the furnace and oil burner used with it.

Lime-Roasting a Galena Concentrate with Reference to the Savelsberg Process. Laboratory Experiments in: Hofman, Reynolds and Wells, *Bull. A. I. M. E.*, 1907, 37. Two converters were used (1) a Morgan clay crucible No. K (2) a small sized (14 in. dia.) detached Devereaux slag-pot. Blast from Roots blower No. ½. Pressure 2 to 10 in. water. A singulo-silicate charge with limestone equal to from 20 to 26 per cent. of weight of ore gives best results.

Roasting of Argentiferous Cobalt-Nickel Arsenides of Temiskaming, Ontario, Canada: Howe, Campbell and Knight. *Bull. A. I. M. E.*, 1907, 53. Ore contained about 56 per cent. arsenic, which can be reduced to 41 per cent. below 700° C. It can be further reduced to 17 per cent. at a temperature above 810°. Smaltite is analogous to pyrite in roasting. Charcoal does not help the roast.

Smelting Cobalt Ores, Method of: H. W. Hixon. *Canadian Mining Inst. 1907: E. and M. J.*, 83, 426. (1) Sorted and metallic silver cupelled. (2) Ball mill and sample. (3) Roast to expel the arsenic and sulphur. (4) Bag house to recover dust and arsenic. (5)

Roasted ore should then be briquetted and smelted in a blast furnace with an equal weight of galena. Base bullion could be sold. Matte and Speiss re-roasted and re-smelted. The second matte to be converted. Cobalt goes into converter slag which would be re-smelted in separate furnace with a portion of the briquetted flue dust from the roasting of the ore. Speiss formed and cobalt extracted chemically.

Treating Cobalt Ores: E. and M. J., 83, 612. The A. S. and R. charges $10 per ton dry weight for treatment and pays for 94 per cent. of silver contents. The Balbach S. and R. Co., charge on a sliding scale $6 per ton on 400 oz. ore to $2 for 700 to 800 oz. ore and pay 93 per cent. of silver contents. The nickel, cobalt and arsenic are not paid for. The Oxford Copper Co. pay for silver and cobalt.

Lithage. On the freezing point of: F. O. Doeltz and Mostowitsch. *Metallurgie*, 1907, 289. The melting point is 906° C. As low as 850° sintering occurs and this as in the case of ZnO, CdO and PbS is due to volatilization below the fusing point.

Lead Oxide, Copper Oxide and Tin Oxide. The Reduction of, by Carbon: Doeltz and Graumann. *Metallurgie*, 1907, 420. Lead oxide begins to be reduced at 530 to 555°, CuO above 500° and SnO_2 above 800° C.

ZINC, CADMIUM.

Zinc: W. R. Ingalls. *Min. Ind.*, XV., 753. Gives production of zinc and zinc oxide. Production by states, etc. Progress of metallurgy of zinc see —

The Metallurgy of Zinc in 1906: W. R. Ingalls. *E. and M. J.*, 83, 20. Takes up the subject of the mechanical separation of mixed sulphide ores. At Broken Hill the flotation processes have proved better than magnetic separation. In roasting the Hegeler furnace is favored. The Imbert process has been tried in France, the Ganelin at Overpelt and the Schmieder process in Silesia.

Roasting for Magnetic Concentration of Zinc Ores: F. H. Trego. *E. and M. J.*, 83, 613. Contrasts the cylindrical furnace with the Trego, a new circular table roaster.

Zinc Smelters in the United States: E. and M. J., 83, 775. Gives a list of the operating plants.

Production and Consumption of Spelter in 1906: W. R. Ingalls. *E. and M. J.*, 83, 937. Spelter produced in 1906 was 225,494 short tons. Deals with new plants to go into operation in 1907 and also the enlargement of the old plants.

Iron Oxide to Zinc Blende. The Behaviour of: C. A. Graumann. *Metallurgie*, 1907, 69. The work was done in a combined carbon tube and Kryptol resistance furnace. In a neutral atmosphere weak reaction. In a slightly oxidizing atmosphere SO_2 and Zn are volatilized, the latter burning. In a reducing atmosphere metallic iron is formed from Fe_2O_3 by CO. The Fe then reduces ZnS. This cannot be used commercially because CO_2 prevents the condensation of metallic zinc. ZnS is reduced to metal by metallic iron in a fine state of subdivision. A temperature of 1300° C. is required.

Zinc Blende and Calamine. The Mixing of: F. Juretzka. *Metallurgie*, 1907, 85. A paper on the production of zinc from mixtures of blende and calamine.

Zinc Oxide. On the Reduction of: F. O. Doeltz and Graumann. *Metallurgie*, 1907. Experiments on the reactions between ZnO and sugar carbon, mixed and heated to 900, 950 and 1000° C. in a platinum resistance furnace. Loss of weight gives zinc reduced. Again ZnO and sugar carbon in different ratios were heated in an atmosphere of nitrogen to 800, 850, 900 and 1000° C. CO_2 and CO determined. Other experiments with zinc oxide and CO.

Theory and Practice of Sherardizing: A. Sand. *Electro. and Met. Ind.*, V., 187. Articles are placed in cast iron drum and covered with commercial zinc dust. Drum closed tight and heated to 300° C. or more. Micrographs show that alloys are formed at the surface.

Cadmium Oxide. On the Reduction of: F. O. Doeltz and Graumann. *Metallurgie*, 1907, 419. Used the same methods as for ZnO.

Zinc Oxide and Zinc-Lead Pigment Manufacture: W. F. Gordon. *E. and M. J.*, 83, 1033. Describes the process used at Coffeyville, Kan. Ores are sulphide, carbonates and silicates of zinc and lead. Roasted in McDougalls. Pigment furnaces. Bag room. Copper, gold, and silver remain in the slag which goes to the blast furnace.

FUELS, FURNACES, ETC.

Water-gas : O. Nagel. *Electro. and Met. Ind.*, V., 19. Translation from Baron H. Jüptner von Jonstorff's Chemical Technology of Energies. Deals with the thermochemistry of the subject with tables and charts.

By-product Coke Ovens in America : E. A. Moore. *Mines and Mins.*, 1907, XXVII., 253. Describes 100-oven plant at Camden, N. J. Methods and costs.

Coke Oven Construction: W. M. Judd. *M. and Mins.*, XXVII., 278. A paper on the beehive oven with illustrations.

The Ziegler Process for Coking Peat: O. K. Zwingenbergen. *E. and M. J.*, 83, 143. The peat is first bricked, then dried and afterwards goes to the retort ovens. The by-products are saved. Six illustrations of the plant at Beuerberg are given.

The Stassano Thermo-Electric Furnace: E. Stassano. *E. and M. J.*, 83, 1,135. Describes the rotary and other forms of this furnace.

The Electric Furnace in Steel Making and Copper Smelting: T. S. Anderson. *E. and M. J.*, 83, 1,251. Describes the two forms of the Anderson furnace.

Laboratory Induction Furnace: Electro. and Met. Ind., V., 151. This Kjellin furnace has a maximum capacity of 15 kw. and is designed for a frequency of 150 to 200 cycles per second. 30 lb. ingots were poured. Furnace tilts just like the full-sized Kjellin. See

Electric Steel: Electro. and Met. Ind., V., 92. An abstract of H. Roechling's paper in Stahl und Eisen of January 16 on the Kjellin furnace at Voelklingen. V. 25 and 58, abstracts of Eichhoff's paper on the Herault process at Remscheid in Stahl und Eisen of January 9.

A 24-ton Induction Furnace for Steel Manufacture: Elect. and Met. Ind., V., 172. Shows illustrations of the tilting Kjellin furnace at Voelklingen.

The Girod Ferro-Alloy Works and the New Girod Steel Furnace: R. S. Hutton. *Electro. and Met. Ind.*, V., 9. The works produce ferro-silicon, ferro-chrome, ferro-tungsten, ferro-molybdenum, ferro-vanadium. Analyses are given. The Girod furnace is shown (*Electro. and Met. Ind.*, II., 309.) Arc type single vertical electrode. Power 300–350 H.P. charge 1.2 to 1.5 tons. Electric resistance furnace used for reheating small steel ingots.

Furnace Output: *Metallurgie*, 1907, 97. A paper giving figures in connection with converters, lead furnaces, copper blast furnaces and reverberatories.

Dust in Flues, Settling and Collection: L. S. Austin. *M. and S. P.*, XCIV., 668. Gives time of fall of particles, etc., and shows plan for flues from copper smelter to stack.

The Location of Smelting Works: H. Lang. *E. and M. J.*, 83, 565. Shows the advantage of the level site over the terraced one. Discusses the plant of the future.

Pyrometry, Metallography.

Acetat Kupfer (the so-called Allotropic Copper): C. Benedicks. *Metallurgie*, 1907, p. 5, 33. The microscope shows that "allotropic" copper as deposited from a neutral acetate solution consists of an intimate mixture of copper oxide and copper. The question arises, do the properties depend upon "allotropic" copper or copper oxide. Under certain conditions "allotropic" copper was produced free from oxide and showed peculiar properties.

Refining and Overpoling Electrolytic Copper. A Study in: Hofman, Hayden and Hallowell. *Bull. A. I. M. E.*, 1907, 275. A study of copper (physical and chemical) during refining in a reverberatory, a study of overpoling electrolytic copper by overpoling tough-pitch copper in crucibles and in the reverberatory. In set-pitch copper the solubility of gas is at a minimum on account of low temperature and high percentage of oxygen. Overpoled has the solubility of gas at a maximum on account of high temperature and entire absence of oxygen.

Copper and Phosphorus: Heyn and Bauer. *Metallurgie*, 1907, 242, 257. Alloys up to 15 per cent. P were made. Copper and Cu_8P (1024° C.) form a eutectic at 8.27 per cent. P and 707° C.; above 14.1 per cent. P mixed crystals of Cu_3P and probably Cu_5P_2 are formed.

Manganese and Copper. The Alloys of: S. Wologdine. *Rev. de Met.*, 1907, 25. The compound Mn_4Cu (1140° C.) and manganese (1275° C.) form a eutectic at 89 per cent. Mn and 1005° C. From 0 to 40 per cent. Mn we have a series of solid solutions, the temperature falling to 845° C. at 40 per cent. Mn. From 40 per cent. Mn to 78 per cent. Mn (Mn_4Cu) the curve shows a uniform rise and denotes the solidification of Mn_4Cu while the residual alloy freezes at a constant temperature, 850° C.

Sulphur, Selenium and Tellurium in Copper. Microchemical determination of. Hinrichsen and Bauer. *Metallurgie*, 1907, 315. Deals with the determination of S, Se and Te by colorimetric methods.

Melting point of Eutectic Mixtures. Variation in the: C. Benedicks and Arpi. *Metallurgie*, 1907, 417. Certain eutectics can be heated

to temperatures above their freezing point without melting due to the coarseness of their grains.

Cadmium-Bismuth Alloys: A. Portevin. *Rev. de Met.*, 1907, 389. Bismuth freezes at 305°, cadmium at 261°. The eutectic occurs at 138 C. and 37 per cent. bismuth.

Tin-Nickel Alloys: L. Guillet. *Rev. de Met.*, 1907, 534. The curve consists of six branches. (1) Aa from the freezing point of tin to the eutectic 0.1 per cent. Ni at 230° C. (2) aB. B is a transition point. (3) BCD. C is a maximum 43.5 per cent. Ni, a solid solution γ. γ + liq. = δ at 790° C. (4) DE corresponding to the solidification of the solid solution β. (5) EF, the freezing of the solid solution α. At E 1150° C. we have the eutectic of α and β from 62.5 to 95 per cent. Ni. At D 1170° C. we have the transformation γ + liq. = β. δ corresponds to NiSn. Both α and β have transformation points in the solid.

Copper, Silver and Lead: Friedrich and Leroux. *Metallurgie*, 1907, 293. The systems PbCu, AgCu and AgPb were redetermined. The ternary eutectic is ½ to 1° lower than that of PbAg and at a composition 0.5 per cent. Cu, 2.0 per cent. Ag and 97.5 per cent. Pb. 302° C.

Nickel and Arsenic: R. Friedrich. *Metallurgie*, 1907, 200. A series up to 57.5 per cent. As were made, their critical points determined and microstructure examined Nickel with 5.5 per cent. As in solid solution and Ni_5As_2 (998° C.) form a eutectic at 898° C. and 27.8 per cent. As. Ni_5As_2 occurs as α and β modifications. Ni_5As_2 and NiAs (968° C.) form a eutetic at 968° C. and 43.3 per cent. As. It is probable that Ni_3As_2 separates out in the solid. The compounds Ni_3As and Ni_2As do not exist.

Iron and Arsenic: K. Friedrich. *Metallurgie*, 1907, 129. A series of alloys from 8 to 56 per cent. As. Fe_2As and Fe_3As_2 are proved; FeAs considered probable. Fe_2As and Fe form a eutectic at 830° C. and 30 per cent. As. Fe_2As forms solid solutions with FeAs when rapidly cooled, when slowly cooled with a second compound. At 800° C. the mixed crystals break up into Fe_2As + Fe_3As_2 or Fe_3As_2 and FeAs as the case may be. With higher percentages Fe_5As_4 is formed. Fe_2As freezes at 919° C. FeAs at 1030°.

Wrought Iron. The Structure of: A. Sauveur. *Electro. and Met. Int.*, V., 119. Shows the structure of wrought iron by the reproduction of the best and most typical drawings and photomicrographs published.

Iron-Carbon Alloys. Present Position of Our Knowledge of the Freezing and Cooling of: P. Goerens. *Metallurgie*, 1907, 137, 173. A continuation of a former paper. Gives the latest equilibrium diagrams of the iron-carbon systems. Very many exceptionally good photographs as illustrations. In the unstable system we have the following sequence: Austenite, martensite, troostite, ormondite, sorbite, lamellar pearlite, granular pearlite. The stable system is austenite, troostite cementite and ferrite with temper-carbon.

Chromium — The Influence of: on the Solubility of Carbon in Iron and the Formation of Graphite: P. Goerens and A. Stadeler. *Metallurgie*,

1907, p. 18. Alloys were made from Swedish iron 4 per cent. C and ferrochrome 67.2 per cent. Cr. Melts were made up to 62 per cent. Cr, the freezing points and critical points determined and the microstructure worked out, which shows that chromium retards the formation of graphite very strongly.

Malleable Cast Iron. Physical Properties and Composition of: F. Wüst. *Metallurgie*, 1907, 45. The strength of malleable is not affected by Si, P or S if below 1.2, 0.1 and 0.2 per cent. respectively. When once the carbide has been split up, the strength is not affected by the oxidation of the temper-carbon. When S is above 0.15 per cent. the ductility is lowered. Further annealing does not affect the strength but may improve toughness.

Chrome-Tungsten Steels: L. Guillet. *Rev. de Met.*, 1907, 5. This series has been worked out as in the case of other alloy steels. The group with sorbite (or troostite) and carbide are best suited for high speed steels.

Ferro-Cuprous Sulphides. The Constitution of: Hofman, Caypless and Harrington. *Bull. A. I. M. E.*, 1907, 25. The freezing point curve falls from 978° C. Cu_2S to 850° C., the eutectic at about 86 per cent. FeS, then shows a very slight rise to 65 per cent. FeS and 860° C. followed by a regular rise to 1152° C. at 100 per cent. FeS. The microscope shows that FeS and Cu_2S form no compound. Eutectic disappears at FeS 20 per cent., Cu_2S 80 per cent. At the FeS side of chart the disappearance of the eutectic is not recorded because it cannot be distinguished from that found in FeS which is composed of FeS, Fe and iron oxide.

Autographic Pyrometer with fixed Photographic Plate: S. Wologdine. *Rev. de Met.*, 1907, 552. The beam of light is thrown from the galvanometer mirror onto a second mirror and thence onto the plate. The second mirror moves on its horizontal axis by means of an arm and plunger in mercury well and thus we get two displacements corresponding to time and temperature.

General.

Metallurgical Calculations: J. W. Richards. *Elect. and Met. Ind.* IV., 486, V.,11, V., 44, V., 79, V., 165, V., 212. The Thermo-Chemistry of the Bessemer Process. The Temperature Increment in the Bessemer Converter. The Open Hearth Process. Thermal Efficiency of Open-Hearth Furnaces. The Electrometallurgy of Iron and Steel. Roasting and Smelting Copper Ores.

Sulphides at a Red Heat. Action of Steam on: A. Gautier. *Bull. Soc. Chim.*, 1906, 35, 934. The action of steam on iron sulphide at incipient read heat is mainly:

$$3FeS + 4H_2O = Fe_3O_4 + 3H_2S + H_2.$$

The action is different in the case of those sulphides which do not decompose steam at red heat

$$3PbS + 2H_2O = 3Pb + 2H_2S + SO_2,$$
$$2H_2S + SO_2 = 2H_2O + 3S,$$
$$PbS + 2SO_2 = PbSO_4 + 2S,$$

which shows how a native metal may result from the sulphide under the action of steam.

$$Cu_2S + 2H_2O = 2Cu + SO_2 + 2H_2.$$

The sulphides of silver yield metallic silver and sulphur dioxide even more readily.

Injury to Vegetation in an Urban Villa near a large Industrial Establishment. Search for the Causes of: P. Frazer. *Bull. A. I. M. E.*, 1907, 377. Bibliography. Gives methods and journal of investigation which proved the oxides of sulphur were present. Amount of soot deposited was determined.

ABSTRACTS OF RECENT MINING DECISIONS.*

LOCATION OF CLAIMS.

RELOCATION.

A relocation of a mining claim can be made only where there has been a prior valid location, and the rights of the original locator have terminated. A relocation is authorized only for forfeiture or abandonment of a prior location. By making a relocation the locator makes admission of the validity of the prior location and precludes himself from contesting it. "Mining claims are not open to relocation until the rights of a former locator have come to an end. A relocator seeks to avail himself of mineral in the public lands which another has discovered. This he cannot do until the discoverer has in law abandoned his claim and left the property open for another to take it up."

A notice given by a relocator is said to be a solemn admission in an instrument which is made a matter of record that such relocator is not a discoverer of mineral, but an appropriator thereof on the ground that the original discoverer had perfected his right. It becomes in a certain sense an instrument of title, a record; and it is the equivalent of an admission of record to the original locator that such relocator claims a forfeiture by reason of a failure of the first locator to make his annual expenditure.

Zeres *v.* Vanina 150 Fed. 564. (January, 1907.)

POSSESSION FOR STATUTORY PERIOD EQUIVALENT TO LOCATION.

The rule has been declared that the working of a mine for the statutory period before the adverse right exists is equivalent to a location under the act of Congress. As said by the Court: "In other words, a party who has done such work occupies the status and possesses the rights of a locator, no more and no less. As in the case of a holder of a valid location he has good title as against all but the government, so long as he does the annual labor." The holder of such a possession, no less than the holder of a location, must possess the necessary qualifications as to citizenship, and he must prove, as well as the locator, the possession of $500 worth of labor or improvements before he can secure a patent, and pending his right to patent, he must perform the annual assessment required of the locator by the statute.

Upton *v.* Santa Rita Min. Co., (N. Mex.) 89 Pac. 275. (February, 1907.)

LOCATION NOTICE ON OVERLAPPING AREA.

A failure to comply substantially with the statute requiring the locator of a mining claim to post a location notice in some conspicuous place

* Prepared for the SCHOOL OF MINES QUARTERLY, by J. W. Thompson, Attorney-at-Law, Indianapolis, Indiana.

on the location, renders the location void. But the mere fact that the portion of the claim upon which such notice was posted by mistake overlapped or conflicted with the previous existing claim, was held not sufficient to invalidate the location. The location was held to be good as to so much of the claim not in conflict.

Upton *v.* Santa Rita Min. Co., (N. Mex.) 89 Pac. 275. (February, 1907.)

DISCOVERY OPENINGS MUST BE CONFINED TO CLAIM — MONTANA STATUTE.

A statute of Montana provides that the locator of a mining claim must sink a discovery shaft on the lode or claim, to the depth of at least ten feet, or deeper if necessary, to show a well-defined crevice or valuable deposit; but it is also provided that a crosscut, which cuts the lode at the depth of ten feet below the surface, is equivalent to the discovery shaft. The statute also provides for the filing for record of a declaratory statement, which, among other things, shall state the dimensions and location of the discovery shaft or its equivalent sunk upon the claim. But under these statutes where it appeared that a crosscut intersected the vein on a claim one hundred and thirty-two feet below the surface, but that the only means of reaching such crosscut was down a shaft located upon another patented claim, it was held that such crosscut, even if made by defendant, did not meet the requirements of the statute for development work, and no other work having been done, there was no valid location of the claim.

Butte Consol. Min. Co. *v.* Barker, (Mont.) 89 Pac. 302. (March, 1907.)

CONFLICTING RIGHTS.

LEGAL SEVERANCE OF MINERALS FROM LAND.

A conveyance by an owner of land of the undivided half interest of the coal under the land, which was executed after a conveyance of the other half of the coal thereunder, operated as a severance of the coal from the land, so that thereafter the mere possession of the surface did not extend to the possession of the coal; and the holder of the paper title to the coal was entitled to recover in ejectment against the owner of the land, unless the latter and those under whom he claimed title had been in the adverse possession of the coal as such, independent of the possession of the surface, for more than ten years before the beginning of the action.

Gordon *v.* Park, (Mo.) 100 S. W. 621. (March, 1907.)

MINING TITLE — RIGHT OF POSSESSION.

In an adverse suit to determine the title to a mining claim the rule governing ordinary ejectment suits, that mere prior possession shall constitute a basis for recovery, does not apply, but the rule is that the defendant, as well as the plaintiff, must recover on the strength of his own title. And the Land Office is not bound to recognize the finding

of a court for a defendant, unless it be upon proof of right of possession; and any local court which proceeds in defiance of this condition simply imposes on the litigants the probable penalty that the Land Office will not regard their conclusions thus reached in disregard of the act of Congress. All the Land Department asks from the court is: "Who, upon the merits, has the right of possession?" All other findings are said to be superfluous and an invasion of the jurisdiction of the Land Office; this is true because "it remains in every case for the Land Department to determine all other questions touching the right of patent."

Upton *v.* Santa Rita Min. Co., (N. Mex.) 89 Pac. 275. (February, 1907.)

ADVERSE CLAIMANTS — CONFLICTING AREAS.

In an action to determine the rights of respective claimants to a mining claim under the United States statutes, the complaint must contain a definite description of the particular area in conflict that might belong to each of the adverse claimants, and the judgment following the pleading must also designate the particular part of the area in conflict belonging to each of the adverse claimants.

Smith *v.* Imperial Copper Co., (Ariz.) 89 Pac. 510. (March, 1907.)

MINING LEASES.

OIL AND GAS LEASE — CANCELLATION.

The lessee of a gas and oil lease failed to drill a well within the time stipulated, and the lessor served notice on the lessee that he would not accept any renewal and demanded that the work be begun and pushed to completion. On failure of the lessee to comply with the lease by drilling wells as required by its terms, the lessor was entitled to a cancellation of the lease.

Puritan Oil Co. *v.* Myers, (Ind.) 80 N. E. 851. (April, 1907.)

GAS AND OIL LEASE — LICENSE TO EXPLORE.

A gas and oil lease conferred upon the lessee the right to enter upon, operate for, and procure oil and gas upon the lands of the lessor. Such a lease was held to grant a license to enter upon the premises described and explore for gas and oil, and in case either or both is found, the right to produce and sever them from the real estate.

Kansas Nat. Gas. Co. *v.* Board of Commrs., (Kans.) 89 Pac. 750. (March, 1907.)

OIL AND GAS WELL — RIGHTS OF TENANTS IN COMMON.

Under the laws of West Virginia a tenant in common has no right either in law or equity to have that part of the estate set off to him on which he had sunk an oil and gas well, for the reason that under the laws of that state the drilling of an oil or gas well is a waste for which one tenant is liable in damages to his co-tenant.

Dangerfield *v.* Caldwell, 191 Fed. 554. (February, 1907.)

REMOVAL OF COAL — LIABILITY FOR REASONABLE ROYALTY.

In an action against an adjoining land owner for damages for taking coal from the plaintiff's land, it was agreed that the defendant should take coal from under the plaintiff's land and give a bond to account therefor in case the plaintiff succeeded in his action. On the trial of the case the boundary was determined in favor of the plaintiff, and it was held under the bond and the agreement that the defendant was only liable to the plaintiff for a reasonable royalty on the amount of coal removed.

Sandy River, etc., Coal Co. *v.* Whitehouse, etc., Coal Co., (Ky.) 101 S. W. 319. (April, 1907.)

MEASURE OF DAMAGES FOR WRONGFUL TAKING OF COAL AND ORE BY MISTAKE.

In an action for damages for unlawful working of a mine and taking coal or ore therefrom, the rule is that if the taking was not a wilful trespass but was the result of an honest mistake as to the true ownership of the mine, the measure of damages is the value of the coal or ore as it was in the mine before it was mined.

Sandy River, etc., Coal Co. *v.* Whitehouse, etc., Coal Co., (Ky.) 101 S. W. 319. (April, 1907.)

LIABILITY OF LESSOR OF MINE FOR INJURY TO SURFACE SOIL.

The lessor of the owner of coal in place removed such coal without leaving any support whatever, and the lessor with knowledge of such fact took no steps to avoid the injury to the surface soil, and received royalties reserved by the lease on all the coal mined. In an action by the owner of the surface land the lessor was liable to him for the injury sustained by reason of the sinking of the surface soil over the mine from which the coal was taken.

Campbell *v.* Louisville Coal Min. Co., (Colo.) 89 Pac. 767. (April, 1907.)

WAIVER OF STATUTORY BOND TO SECURE SURFACE OWNER — NO RELEASE.

The waiver, on the part of the surface owner, of the statutory bond to secure the payment of damages to the surface soil resulting from the removal of the minerals, does not release the operator of the mine from damages to the surface soil that may be occasioned by removing the minerals. The bond is for the benefit of the surface owner, the giving of which is a condition precedent to the removal of the minerals, but may be waived.

Campbell *v.* Louisville Coal Min. Co., (Colo.) 89 Pac. 767. (April, 1907.)

LIABILITY FOR NEGLIGENCE TO SUPPORT SURFACE.

Where the surface of land belongs to one person, and the coal thereunder to another, the owner of such coal cannot remove it without

leaving support sufficient to maintain the surface in its natural state; and the lessor of such coal is liable to the owner of the surface soil for any damages occasioned by the negligence of the lessee in failing to properly support and protect the surface. Every such lease contains an implied covenant that the lessor would so conduct the mining operation as not to damage the surface by undermining, and with the right to remove only so much coal as it could get without injury to the surface.

Campbell *v.* Louisville Coal Min. Co., (Colo.) 89 Pac. 767. (April, 1907.)

Title to Mineral.

The lessee of a mine has no title to any specific portion thereof, until mineral of the kind described in the lease is actually produced and severed, so that it becomes personal property. The legal title to and the possession of the entire mass and volume remain in the owner of the strata in which it is confined until so severed.

Kansas Nat. Gas Co. *v.* Board of Commrs., (Kans.) 89 Pac. 750. (March, 1907.)

Minerals in Place — Taxation.

There can be no taxation of separately owned mineral rights, except when the right or title to minerals in place has been severed from the right or title to the remainder of the land, and has become vested in a person other than the one having the right or title to the remainder of such land. When the statute applies, the right or title to minerals is taxed as realty.

Kansas Nat. Gas Co. *v.* Board of Commrs., (Kans.) 89 Pac. 750. (March, 1907.)

ASSESSMENT WORK.

Proof of Assessment Work.

A forfeiture of a mining claim for failure to do annual work can be established only upon clear and convincing proof of such default proved against the former owner by the party assailing his title. But where the owner has failed to make and file the proof of labor required by statute, such owner was held not entitled to invoke this rule, for the reason that the statute places the burden of proof upon him to show that such work has been done according to law, and not upon his opponent to show that it has not been done.

Upton *v.* Santa Rita Min. Co., (N. Mex.) 89 Pac. 275. (February, 1907.)

LIABILITY OF INJURY TO MINER.

Admissable Evidence in Case of Accidents.

In order to entitle an employe to recover for injuries received by the falling of the roof of an entry to the mine of his employer, it is incumbent upon him to show that the employer was negligent in using props in the manner he did, and he may show another safe and convenient way of supporting such roof. And where the employer offered evidence of a custom of supporting similar entries, it was held competent for the

employe in rebuttal to show how the roof of an entry should be timbered.

Madden *v.* Saylor Coal Co., (Iowa) 111 N. W. 57. (March, 1907.)

CONSTITUTIONALITY OF STATUTE IMPOSING RESPONSIBILITY ON MINE OWNERS.

The liability imposed upon mine owners by the Illinois mining act, of responsibility for the defaults of mine managers and mine examiners, who are required by that act to be selected by the mine owners from those holding licenses issued by the State mining board, created by such act, does not deprive them of their property without due process of law, where it is not obligatory upon any mine owner to select a particular mine examiner, or to retain one after selection, if found to be incompetent.

The statute selecting mine owners as a class upon which to impose liability and responsibility for the defaults of mine managers and mine examiners required to be selected from those holding licenses issued by the State Mining Board, is not repugnant to the constitution as denying the equal protection of the law.

Wilmington Star Min. Co. *v.* Fulton, 21 Sup. Ct. 412. (March, 1907.)

RISKS ASSUMED BY MINER.

A miner whether working as a servant of the mine operator, or working in the mine by invitation, is held to assume the risks incident to such work; but increased risks caused by the negligence of the operator are not regarded as incident to the business, and are not assumed by the miner.

Birmingham Min. etc., Co. *v.* Skelton, (Ala.) 43 So. 110. (February, 1907.)

EMPLOYERS LIABILITY — ACCUMULATION OF GASES.

A mine owner is not relieved from liability for his neglect to perform a statutory duty to prevent the accumulation of dangerous gases, which led to an explosion, because there was another efficient cause contributing to the accident for which a recovery could not, under the declaration, be had.

Wilmington Star Min. Co. *v.* Fulton, 21 Sup. Ct. 412. (March, 1907.)

APPLIANCES FOR ENTERING AND LEAVING MINE.

Where the mine boss, the blacksmith, and two miners occasionally rode in a car used for hoisting ore, and where the mine boss directed miners not to use the car, and where the mine was equipped with a system of ladders, well lighted, kept in good order, and commonly used by the mine employes in going to and from the mine, it was held that such car was not an appliance furnished by the master to be used as a passenger elevator.

Burns *v.* Old Sterling, etc., Min. Co., (N. Y.) 80 N. E. 927. (April, 1907.)

INJURY TO MINER — RECOVERY — KNOWLEDGE OF DANGER.

A miner may recover for injuries received from the falling of the roof of a mine, unless he knew of the danger, or could have known of it by the exercise of ordinary care in the course of his employment.

Big Hill Coal Co. *v.* Abney, (Ky.) 101 S. W. 394. (April, 1907.)

MISCELLANEOUS.

POLLUTION OF STREAMS.

The owner of a mine is liable to the owner of property on a stream below the mine for damages which result from acidulated and offensive water pumped from such mine and thrown into the stream, although such disposal of the water was necessary to the operation of the mine.

Bowling Coal Co. *v.* Ruffner, (Tenn.) 100 S. W 116. (February, 1907.)

ADMISSION OF EVIDENCE AS TO ADVERSE CLAIM.

In an action in support of an adverse to a mining claim, where a declaratory statement was introduced in evidence before any testimony had been given, showing that the steps necessary to be taken before a declaratory statement could be filed, had in fact been taken by the locator, and the statement having been followed by the necessary proof, it was held that the admission of such declaratory statement out of its order was immaterial and without prejudice. And the admission in evidence of an original declaratory statement, defective in that it failed to sufficiently describe a discovery shaft, was admissible in evidence, in connection with the amended declaratory statement, as showing an attempt on the part of the locator to comply with the law and the identity of the claim described in the two statements.

Butte Consol. Min. Co. *v.* Barker, (Mont.) 89 Pac. 302. (March, 1907.)

INTENTIONAL VIOLATION OF STATUTE AGAINST INJURING OR ENTERING MINE.

In a prosecution against miners for "intentionally" entering a mine against caution and for injuring the shaft of the mine, it was held that the word "intentional," as used in the statute, only required that the act constituting the alleged violation of the statute should be "knowingly" done.

Koppla *v.* State, (Wyo.) 89 Pac. 576. (April, 1907.)

INJUNCTION PENDING PROCEEDINGS — THREATENED ASSAULT.

An injunction may be granted to prevent a threatened assault to a claimant and his employes, in case they appeared upon or undertook to do any work upon a claim, pending an action to determine the right of possession and ownership of certain mining ground. Such injunction may prevent the defendant, his agents and employes, from in any manner interfering with the plaintiff and his employes, in performing neces-

sary discovery work and other work upon such claim necessary to hold the same, although all the allegations of the plaintiff's complaint were denied by the defendant

Safford *v.* Fleming, (Idaho) 89 Pac. 827. (April, 1907).

INJUNCTIVE RELIEF LIBERALLY GRANTED.

The courts are liberal in granting injunctive relief in mining litigation, in order that neither party may gain any advantage over the other, during such litigation, by force or violation.

Safford *v.* Fleming, (Idaho) 89 Pac. 827. (April, 1907.)

ANNUAL MEETING OF ALUMNI ASSOCIATION.

REPORTS OF OFFICERS OF ALUMNI ASSOCIATION SUBMITTED AT ANNUAL MEETING, HELD AT COLUMBIA UNIVERSITY, APRIL 25, 1907.

REPORT OF TREASURER.

Receipts.

Balance on hand, April 23, 1906:		
Irving Savings Institution	$584.12	
Market and Fulton National Bank	954.19	$1,538.31
Dues collected and remitted by R. E. Mayer		2,173.25
Interest on deposit, Irving Savings Institution		23.58
		$3,735.14

Disbursements.

Collation, annual meeting	$ 204.75	
Alumni Council, dues '06, '07 and acc.	150.00	
School of Mines Quarterly	1,014.92	
Printing and stationary	169.00	
Typewriting, etc.	39.00	1,577.67
Balance on hand		$2,157.47

WILLIAM FELLOWES MORGAN,
Treasurer.

April 25, 1907.

REPORT OF BOARD OF MANAGERS.

On behalf of the Board of Managers the secretary submitted the following:

At a regular meeting of the Board of Managers held at the Columbia University Club on April 10, the following action was taken:

Resolved, That the secretary be authorized to accept the offer of the *Columbia University Quarterly* to send the *University Quarterly* to all members of the Alumni Association of the Schools of Science at a rate of $250 per 1,000 subscriptions.

Resolved, That the secretary be authorized to expend the necessary amount required to pay for the luncheons of all members of the association who may attend the alumni luncheon served on commencement day.

Respectfully submitted,
R. E. MAYER,
For Board of Managers.

REPORT OF SECRETARY.

Total membership, April 26, 1906		928
Class 1906	100	
All others	15	
	115	

Died	9	
Resigned	4	
Dropped	10	
	23	
Net Increase		92
Total membership		1,020

R. E. MAYER,
Secretary.

April 25, 1907.

SUMMARY OF THE ACCOUNT OF THE BUSINESS MANAGER OF THE SCHOOL OF MINES QUARTERLY, COLUMBIA UNIVERSITY. JANUARY 1, 1906, TO JANUARY 1, 1907.

	Disbursements.	Receipts.
Balance on hand January 1, 1906	$	$1,112.84
Alumni subscriptions		1,014.92
Other subscriptions		134.34
Advertisements	60.31	492.08
Back numbers and reprints		312.86
Exchanges	4.00	106.04
Editorial	450.00	
Illustrations	371.68	155.75
Printing	1,003.36	
Postage	96.79	
Addressing, labor and express	41.70	
Stenographer for Ostwald lectures	104.75	
Sundries	13.45	
Interest on bank deposits		16.13
Balance on hand January 1, 1907	1,198.93	
	$3,344.96	$3,344.96

T. H. HARRINGTON,
Business Manager.

January 1, 1907.

UNIVERSITY NEWS.

DEPARTMENT OF CHEMISTRY.

During the past summer session all of the courses offered in chemistry were attended by increased numbers of students, and the total registration in chemistry reached 259, an increase of about 60 per cent. over that of any previous year.

With the opening of the present year, Professor Wells returned from leave of absence.

Dr. K. G. Falk resigned the position of assistant in physical chemistry in order to accept a tutorship in physics, and Mr. Eric Higgins (B.Sc., Victoria University, Manchester) has been appointed to the vacant assistantship.

No other changes in the staff have occurred since the notice in the July QUARTERLY, but some rearrangement of instruction in qualitative analysis has been made in connection with the adoption of a uniform curriculum for the first year in applied science, so that now the first year class is divided into two main sections, each of which occupies the laboratory during one half-year; while for classroom work, each half of the class is divided into two sections in lectures and six in recitations.

The number of students enrolled for organic chemistry is larger this fall than ever before, and the analytical laboratories, both qualitative and quantitative, continue to be overcrowded.

Dr. H. K. Benson, university fellow in chemistry, 1906–07, received the degree of Ph.D. (physical chemistry) in June, and has returned to the University of Washington where he is acting head of the Department of Chemistry.

Of the students who specialized in organic chemsitry last year, Dr. J. M. Nelson is now an instructor at the Rensselaer Polytechnic Institute; Dr. R. R. Renshaw is a member of the chemical staff of Wesleyan University; Dr. William Klaber is with the Chrome Tanning Company at Camden, N. J.; Mr. Schaeffer is with the Schaeffer Alkaloid Works, Maywood, N. J.; Mr. Fisher is in the employ of the Casein Company of America; Mr. Kroppf has returned to the laboratory as a candidate for the Ph.D. degree.

Dr Alfred Hoffman, having completed his work at Berlin University, is now conducting special investigations in the Organic Laboratory.

In a recent examination held by the U. S. Department of Agriculture for the appointment of food inspection chemists, and taken by hundreds of chemists in all parts of the country, it is reported that Dr. H. A. Seil (Ph.D. in organic chemistry, Columbia, 1906) received a a ranking above all his competitors.

Of the students whose major work was in analytical or food chemistry, Dr. L. J. Cohen was offered positions both in the National Bureau

of Standards and in the Bureau of Chemistry of the U. S. Department of Agriculture, but declined in order to accept an appointment in the Department of Chemistry in the College of the City of New York; Dr. N. E. Goldthwaite has joined the staff of the Rockefeller Institute for Medical Research; Mr. Kress has become tutor, and Messrs. Sinclair and Taylor, assistants in this department; Mr. Williams, instructor in Teachers College; Mr. Whitman, lecturer in chemistry in Extension Teaching; Mr. Steel, assistant in physiological chemistry; Miss Rose, lecturer in Cornell University; Miss Stanley, instructor in the University of Missouri; Mr Hinkel has entered the analytical laboratories of Bendiner and Schlesinger, New York City.

In the electrochemical laboratory the space provided for electric furnace work has been doubled. A 25 K.W. auto-transformer built at the university, and a large impedence coil for use with arc furnace have been added to the equipment, and in addition the laboratory has under construction or about to arrive steel pressure and vacuum furnace, induction furnace, pressure pump, Tyrke vacuum oil pump, and direct current motors for stirring purposes.

The 36th general meeting of the American Chemical Society was held at the University of Toronto, June 27–29, under the presidency of Professor Bogert, who, during his stay in Toronto, was the guest of the venerable Professor Goldwin Smith. Papers by Dr. Chambers and Professor Bogert were presented at the meeting.

Dr. Chambers acted as supervisor of examinations at Columbia for the College Entrance Examination Board and as Chief Reader in Chemistry at the June examinations. On July 25 he gave a public lecture on "The Laboratory *vs.* the Plant" before the students of the summer session.

C. F. C.

DEPARTMENT OF CIVIL ENGINEERING.

Some changes have occurred in the instructing staff during the recent summer vacation due to the resignation of Mr. Yates and of Mr. Sewall. Mr. Yates' position as tutor has been filled by Mr. Morrison, who was advanced from an assistantship. Mr. F. S. Foote, Jr., class of 1904, has been appointed as assistant to Professor Lovell, taking the place made vacant by Mr. Sewall; while Mr. D. Gorren, class of 1907, is acting as assistant to Professor Burr.

Due to the change in the arrangements, the testing laboratory, under the immediate supervision of Professor Woolson and Mr. MacGregor, his assistant, has been transferred to the Department of Civil Engineering. Efforts are being made to secure funds through voluntary contributions for increasing and developing this branch of the department work.

The registration for all courses in the department, whether those given to the civil engineers alone, or to the applied science men in general, show a marked increase over the same figures of last year. At Camp Columbia also the increase in the number of students in residence at any one time was apparent. This phase of the university work was under the direction of Professor Lovell with Mr. Sewall acting as his principal assistant.

The work of the summer session as given by the department, at the university, consisted of two courses, one on structures and structural mechanics, and the other on the subject of hydraulics. Advantage of these courses was taken by students. principally from other institutions, and the indications are that the future will still more clearly demonstrate their popularity.

C. E. M.

DEPARTMENT OF ELECTRICAL ENGINEERING.

Messrs. E. S. Holcombe, B.S., lecturer, O. von Dannenberg, B.E., tutor, and S. S. Watkins, E.E., assistant, have resigned their positions in the Electrical Engineering Department. Edgar S. Downs, Ph.D., Victor Greiff, E.E., and S. F. Bond, E.E., have been appointed respectively as tutor and assistants in electrical engineering.

The standardization rules of the American Institute of Electrical Engineers have recently been completed by the committee of which Professor F. B. Crocker is chairman, and have been adopted by the Institute. This work has been going on for two years, and the rules in their revised form are the permanent basis upon which all electrical machinery and apparatus are rated and tested.

Professor Crocker has also been appointed secretary of the American section of the International Electrotechnic Commission, which includes representatives from twelve of the principal countries. This commission has undertaken the universal standardization of electrical machinery and apparatus, which is an interesting and important step in international affairs.

Professor G. F. Sever has been rearranging and developing the Instrument Laboratory during the summer.

Morton Arendt, E.E., instructor, spent two months in Europe visiting various technical schools and electrical establishments.

The gift of a 40-kilowatt motor generator set has been received from the Interborough Rapid Transit Co. through Mr. August Belmont. This is an important addition to the facilities of the department. The department has purchased much new apparatus, for its laboratories, which is available for student use.

The alternating current laboratory has been greatly improved by removing the partitions between two of the rooms, thereby forming one large laboratory, with much improved light and ventilation, as well as facilitating the supervision. The apparatus has been re-arranged and a new switchboard erected by which the various machines may be controlled.

The Electrical Engineering Society of Columbia University had a very successful session during the past college year, with regular fortnightly lectures by prominent engineers. This society has already arranged a similar program for the coming year, and notices of these meetings will be regularly announced in the *Electrical World.*

The introduction of lecture and laboratory instruction in hydraulics, to the course of electrical engineering, which was made last year, has been a valuable addition, and will be continued in the future.

On account of the number of students in the courses of mechanical and electrical engineering, and because of the somewhat different requirements for them, it has been found advantageous to give separate instruction to the fourth-year classes. This division of the classes necessitates partial duplication of the lectures, but is considered well worth the extra work involved.

F. B. C.

DEPARTMENT OF GEOLOGY.

The summer field session with the School of Mines and graduate students was held at High Falls, N. Y., the week before commencement. The party numbered forty and worked upon the geological section to be crossed by the new aqueduct from the Catskills to New York. The strata exhibit many instructive features and the work was of added interest because of the great engineering problem associated with it. In the regular summer session of the university, Professor Grabau gave courses in general geology, comparative geology of North America and index fossils. He was assisted by Miss Elvira Wood, of the United States Geological Survey. All the courses were well attended.

During the summer Professor Kemp spent five weeks in the field for the State Geological Survey, completing the maps of the vicinity of Port Henry, the most important iron mining center of the Adirondacks. Data for an interesting paper upon these huge masses of iron ore were accumulated.

Previously to the opening of the summer session, Professor Grabau visited Letchworth Park, the area recently given the state along the Genesee River, and in coöperation with Mr. E. H. Hall, secretary of the Scenic and Historic Preservation Society established several monuments for determining the recession of the falls. He devoted some time to the investigation of the physiographic problems in this portion of the state accumulating data for a paper on the ancient drainage of this region. Following the summer instruction at the university, he gave his students a trip of four days in the Hudson Valley with headquarters at Hudson, and then went to Boston to attend the session of the Seventh International Congress of Zoölogy, acting as secretary of the Section of Palæozoölogy and presenting two papers. The department work was represented in the exhibit made at the congress by a case containing Devonic specimens so arranged as to show the various lines of parallel evolution which they underwent in mid-Devonic time. During the New York week of the congress, Professor Grabau served on the local committee. He next went to Detroit where in company with Professor Scherzer of the Michigan Survey, he investigated the interesting problems presented by the late Siluric formations and their contact with the Devonic. The results of this investigation will be published during the winter. After that Professor Grabau went two weeks in the northern portion of the lower peninsula, making additional observations and collections for his monograph of the various groups of Michigan. A large collection of fossils was brought back as the result of the summer's work. During the spring Professor Grabau completed a

paper describing the Upper Siluric fossils obtained in the salt shaft of Detroit, Michigan The illustrations for this paper were made by Miss Johnson who for a number of years has been employed as illustrator in the paleontologic laboratory. The paper is now in course of publication by the Michigan State Survey. The third paper in the series of "Studies on Gastropoda" contributed by Professor Grabau to the *American Naturalist* has been published in the October number

Several important papers on the developmental stages in Palæozoic corals were published during the year as the result of work carried on in the paleontologic laboratory, by Messrs. Gordon, Brown and Anderson.

Dr. Berkey continued his work for the State Survey in the Highlands of the Hudson, both in June and September. During July and August he went to Arizona with a small party of School of Mines students for their field work under the department. The mining camp of Tombstone was studied in some detail and comparisons were made with Bisbee.

Dr. Ogilvie passed a month in the field in the southern Adirondacks accumulating notes for a further contribution upon this region.

Dr. Julien during the summer has completed an investigation upon the formation of serpentine and brucite, which has been some years in progress. He also discovered some interesting chipped stones or artefacts in the gravels of Cape Cod.

Mr. T. C. Brown, for two years past assistant in the department, was appointed in June to the staff of the department of vertebrate paleontology of the American Museum of Natural History. He is succeeded by Mr. Edwin Kirk, '07, College. Mr. F. C. Lincoln, fellow in geology, the past year has been appointed professor of geology and mineralogy in the School of Mines, Butte, Mont. Mr. J. E. Hyde, fellow-elect, was employed by the Ohio State Geological Survey during the summer.

The department has received several gifts of specimens from its former students, of which may be specially mentioned a fine suite of ores and fossils from San Pedro, N. M., collected by R. V. Mills, '06, Mines ; and a complete series of rocks and by-products from the Kimberley diamond mines, South Africa, secured through the kind interest of Harold T. Dickinson, '00, Mines. A series of twelve models of the enlarged protoconchs and early conch stages of Tertiary and recent gastropods has been prepared under the direction of Professor Grabau, and plaster casts of these have been made by the American Museum, a duplicate set being placed in the Paleontologic collection of the department, where they constitute a special exhibit. J. F. K.

DEPARTMENT OF MECHANICAL ENGINEERING.

The personnel of the department has changed considerably since last year. Professor F. R. Hutton, Mr. C. T. Swart, Mr. R. M. Strong and Mr. S. H. Woods, all being absent this year, through resignation and expiration of term. The new appointments to the department include Mr. H. L. Parr, instructor, and Mr. E. D. Thurston, Jr., assistant, in the laboratory, and M. E. J. Kunze, tutor in the draughting room. To eliminate duplication, all of the courses given in the depart-

ment on materials, including the testing laboratory, have been transferred to the department of civil engineering, together with the lectures in theoretical hydraulics, and the civil engineering department has transferred to this department its courses on pumping engines, kinematics, hydraulic laboratory, and heating and ventilation.

During the past summer the department offered three courses in the summer school, one on gas engines, one on experimental investigations and one on hydraulic laboratory practice, the students in which, while not very numerous, were of a rather high class, most of them being practicing engineers.

During the coming year it is hoped that some few changes in the course of study will be possible, with a resultant increase in the efficiency of the instruction and a considerable benefit to the students through a readjustment of the time and effort, which at the present time, is very unequally divided, and the work being unduly hard in the third year and considerably too light in the second year. Several special investigations are under way in the department this year, the principal one being concerned with heat transfer through tubes and the conditions affecting it, such as smoothness of surface, the material cast iron, steel and brass, the form of surface, straight, corrugated, webbed and spiked, the substance in contact, steam of high and low pressure, wet, dry and superheated, water, air, and flue gases in as many combinations as possible. The work on the possibilities of denatured alcohol in standard gasoline and kerosene engines for the U. S. Department of Agriculture has been completed and two bulletins have been issued, the first intended for non-technical readers and the second for engineers, containing the detailed test results.

Late in the spring a committee comprised of two members of each class graduated in mechanical engineering met to consider the advisability of organizing, and as the opinions presented were unanimously in favor of it, it is likely that such an organization will be perfected in the fall. C. E. L.

SUMMER SCHOOL OF MINING.

In the past summer twenty-six students were engaged in work or study in the field, for periods ranging from two to nearly three months. They were unaccompanied by instructors from the department. The usual arrangements were made in advance, insuring that the necessary privileges would be accorded by the mine managements for carrying on the course of study, and each student was provided with printed instructions covering fully the required work. All report that they were kindly received and given every opportunity by managers and superintendents.

The larger part of the class was divided into squads of from two to four men, as follows: Five students visited Butte, Montana; K. E. Lloyd and P. L. Roberts at the Stewart and West Stewart mines of Senator W. A. Clark, together with the Gagnon, Parrot and others of the district; and M. L. Hurley, F. J. Huntzicker and E. B. McCartney, at the Anaconda, Pennsylvania, Silver Bow, Berkeley and other properties of the Amalgamated Copper Co. Several of the neighboring mills and smelters were also studied.

O. K. Doty, H. R. Graham, W. T. Miller and R. M. Perry spent about six weeks at the Vindicator gold mine, in the Cripple Creek district, of Colorado, after which they visited several other mines, mills and metallurgical works in the vicinity. A short trip to Leadville was then made, some members of the squad also going to the Telluride district.

G. W. Daniel, F. G. Downs, M. T. Hoster and A. H. Hubbell, had excellent opportunities for study for six weeks in the group of iron mines operated by the Penn Iron Mining Co., Vulcan and Norway, Michigan, of which Mr. William Kelly (class of '77) is General Manager. They afterward visited the Chapin, Currie and Republic iron mines, the Champion copper mine, and several mills and surface plants in the Lake copper region.

R. G. Estee and G. W. Danehy, together with Kingdon Gould, of the third class, accompanied by Frederick Kruse (S. of M., '07) as private instructor, visited mines at Tombstone and Bisbee, Arizona, and Aspen and Cripple Creek, Colorado.

G. M. Richards, J. J. Wolfersperger and C. L. Bruns, Jr., visited the Center Star and other mines in British Columbia. H. A. Stockley and R. K. Brewster studied at the Copper Range, Winona and other mines in the Lake Superior copper district.

Independent trips were made also by O. M. Lewyn and C. J. Cohn; the former to the Wolftone and Coronado mines of the Western Mining Co., of Leadville, together with other properties of the district, the latter studying in several iron mines in the vicinity of Dover, N. J.

Three students obtained positions, or studied, as follows: L. M. Jones served as assistant engineer with C. M. Haight (S. of M., '06), at the Michigan Copper Mine, Rockland, Michigan; J. N. Chrystie, at the Kerr Lake and other mines in the Cobalt, Ontario, district; and P. von Saltza, at the Fayal Iron mine, Eveleth, Minnesota, and a couple of the Lake copper mines.

As evidenced by the note-books and sketches brought back by the members of the class, most of the summer's work has been done in a highly satisfactory manner.

R. P

BOOK REVIEWS.

Shaft Sinking under Difficult Conditions. By J. RIEMER, translated from the German by C. R. CORNING and ROBERT PEELE. Wiley & Sons, New York, 1907. 8vo, pp. xiii and 176, 19 folding plates. Price $3.00.

The author of this book, written in 1905, is connected with the firm of Haniel and Lueg, engineers and contractors, of Düsseldorf, Germany. The book is based on a series of articles and monographs on shaft sinking by different engineers, members of the firm and others, which have appeared at intervals since 1884.

By reason of his connection with this firm Mr. Riemer writes with the authority of a specialist, thoroughly familiar with his subject, but at the same time he identifies himself fully with the interests of the firm, and patents controlled by them are given due prominence. Like many of the trade publications of manufacturers of mining machinery in this country, the book is a valuable contribution to technical literature, and the translators have earned the gratitude of mining men for their unselfish labor in making it known to us in English form.

The "difficult conditions" are water-bearing strata or incoherent material, or both, and the methods of shaft sinking described are: 1, ordinary methods of excavation; 2, boring under water (Kind-Chaudron); 3, the Poetsch freezing method; and 4, "drop-shafts." That the subject is an important one is evidenced by two tables with details of 79 shafts sunk by the Kind-Chaudron (1852–1904) and of 64 shafts sunk by the Poetsch freezing process (1883–1904). These of course represent a small fraction of the many shafts sunk under "difficult conditions."

The great majority of shafts are sunk by ordinary methods but the author devotes but little space to this phase of difficult shaft work though he notices briefly recent improvements tending to facilitate sinking when large amounts of water or loose ground are encountered. Numerous cases are cited of shafts in which five to ten tons of water per minute were encountered, and a few cases in which the water was fifteen, twenty, and even forty tons per minute. (A large direct acting sinking pump of the usual type will raise say 500 gallons or about two tons per minute.)

It is well known that the cheapest method of raising large quantities of water is by hoisting, but the application of this method to shaft sinking is comparatively new. In Tomson's water-hoisting apparatus the buckets are filled in large suspended tanks near the bottom of the shaft fed from the sump below by attached duplex pumps. A plant of this character is described having a capacity of 15 to 16 tons of water (4,000 gallons) per minute at a depth of 800 feet.

Sinking in water-bearing strata or difficult ground is greatly facilitated by building a water-tight cast-iron lining from above downward so that

the water is promptly cut off, expensive temporary lining saved, and the shaft made secure at all times to within a few feet of the bottom. The author describes the application of this method to several shafts and mentions its use in a number of other cases, including one shaft sunk in dry sand.

In heavily water-bearing ground, while it is possible to raise large amounts of water by hoisting or pumping, 40 tons (10,000 gallons) per minute and more, one tenth this amount of water greatly increases the cost of sinking, and may make it best to use one of the special methods, particularly if the shaft be deep, or the ground difficult.

The author devotes 84 pages, nearly half the book, to the Kind-Chaudron system, and gives several interesting examples of the application of this method to sinkings of unusual difficulty, and to shafts in which other methods of sinking had been tried and had failed. This is perhaps the most valuable part of the book, and well repays careful reading. At the end of the chapter the author discusses the application of the method to shafts of greater depth than 400 meters (1,312 feet) and of larger diameter than 15 feet, the present practical limits, and describes his own plans for the solution of these problems.

Chapter III., of 44 pages, contains a very satisfactory description of the Poetsch freezing process with examples from actual practice The author discusses the effect of pressure on frozen ground, and gives his reasons for believing that the process will not be found applicable for great depths. The deepest shaft as yet planned to be sunk by freezing is less than 800 feet. The author states that "the possibility of leakage of the brine because of defective freezing pipes is one of the most serious contingencies to be feared, because any portion of the ground once saturated with the refrigerating solution cannot be frozen, and dangerous inrushes of the surrounding soil during the operation of sinking become inevitable." The contraction of the pipes caused by the cold solution is a frequent cause of breakage, and attempts to introduce expansion joints (or rather contraction joints) have not proved successful. The pipes must then be of the best material, sufficiently heavy and put together with the greatest care. "Projects to do away altogether with the freezing solution . . . have thus far been unsatisfactory in practice."

"Another difficulty inherent in the freezing process is the tendency of all bore-holes to deviate from their initial direction. This tendency increases with the depth of the hole and to it are chiefly due the suggestions which have been advanced for freezing the shaft in sections." Freezing in sections occasions delay and extra expense, and the author believes that it will prove "a makeshift to be adopted only when it cannot be avoided." At the end of the chapter is a table giving details of 64 shafts sunk by this method between 1883 and 1904.

Chapter IV. deals with the well known method of sinking through quicksand and other incoherent material by the use of drop shafts. Like the chapter on sinking by hand this one is short, 34 pages only. In recent years this method has been greatly modified and improved, and shafts of large diameter are now successfully sunk through great depths of difficult ground. It has not been found possible, however, to

avoid the tendency of the loose ground to break into the bottom of the shaft and thereby create large cavities outside of the sinking lining. The subsequent fall of large masses of ground in these cavities not infrequently bends or cracks the heavy iron linings used, and at times has completely wrecked the shaft. The author states, page 142, "of all the drop shafts I have seen only two wholly escaped deformation. . . . This statement holds good for both iron and masonry shafts."

To guard as far as possible against these inrushes of sand, and to avoid the consequent formation of dangerous cavities outside the sinking lining, drop shafts are now in the majority of cases kept full of water, and excavation made by dredging or boring. If the shaft is to be pumped out the bottom is sealed beforehand with a thick plug of concrete. Neglect of this precaution has been the cause of many expensive accidents to the shaft linings. In sinking, the cutting edge of the lining is kept in advance of the excavation by the use of powerful hydraulic jacks. To lessen the danger of crushing, the iron linings are strengthened by broad rings between the segments, forming horizontal flanges or ribs, with a heavy concrete lining resting on these rings.

The descent of shaft linings is opposed by friction, and sooner or later they stick fast. The first lining is usually of thick masonry, and is made large enough to permit the use of several iron, or iron and concrete linings, telescoping inside each other. These are forced down in turn by powerful hydraulic jacks, until the desired depth is reached. These hydraulic jacks are placed under a massive iron ring inside the first heavy masonry lining, which ring in turn is held in place by bolts extending to the shoe, so that the whole weight of the upper lining is available for forcing down the others. By this means a single lining has been pushed down 360 feet, though frequently they stick at 100 feet and less. In more solid ground the use of a trepan, similar to those used in the Kind-Chaudron method, may be required. With tools of this type there is less tendency for the shaft to diverge from the vertical than when sack borers or other rotary cutters are used. The chapter concludes with the description in detail of the sinking of a number of shafts, successful and unsuccessful. Among others the Adolf Shaft, near Alsberg (1900–1904), is of interest. A heavy masonry shaft was sunk to 105 feet. Inside of this two telescopic iron linings were sunk to 252 feet and 426 feet respectively. In forcing down the last lining the attempt was made to lessen the friction of sinking by the use of water pumped under pressure into the ground outside of the shaft through a number of small holes 23 feet above the cutting edge. This proved eminently successful, the shaft lining sinking of its own weight, though just before the water was used, it had failed to sink under a pressure of 3,000 tons. It must be added, however, that the shaft lining subsequently proved to be badly damaged by falls of ground outside, showing that large and dangerous cavities had been formed, possibly by the water used to lessen friction. The idea of using water for this purpose was suggested by the ease with which the linings were sunk at three shafts where there was a natural artesian flow of water outside the shaft lining, overflowing at the surface at the rate of about 24 gallons per minute. It is interesting to note that one of these three

shafts (Hugo No. 1) was completely wrecked by falls of ground and had to be abandoned.

The book is illustrated by 18 large folding plates, and an equal number of illustrations in the text.

H. S. M.

The Copper Mines of the World. By WALTER HARVEY WEED, Geologist, U. S. Geological Survey, 1883–1906; Fellow Geological Society of America; Member Institution of Mining and Metallurgy, etc. New York, Hill Publishing Co., 1907. Pp. xiv + 375, 159 illustrations. Price $4.00.

This book appears opportunely at a period of exceptional interest in the copper mining industry and the market for the metal. It deals chiefly with the copper deposits themselves; their geography, genesis and geological occurrence, together with the mineralogical and chemical characteristics of the ores. In fact, the author gives more attention to the economic geology than to the mining of copper. The methods of mining adopted in the prominent mines and districts are briefly outlined, the descriptions being accompanied by numerous illustrations of specific cases, showing the shape and associations of the veins or orebodies. With the purpose of keeping the subject matter within a reasonable compass, the author omits to treat of the metallurgy of copper.

In Chapter I. the general distribution of copper deposits is outlined. Chapter II. contains tabulated statements of the world's production of copper and the range of prices; also some considerations relating to the future supply and demand, with a few figures of cost of mining and value of ores Chapter III. treats of the copper-bearing minerals, associated gangue minerals and deleterious constituents of copper ores. In Chapters IV. and V., covering some 16 pages, a brief account is given of the copper outcrops and gossan formation, and the facts concerning secondary sulphide enrichment. Chapter VI., containing a discussion of the genesis of copper deposits, is in many respects the most interesting in the book, relating as it does to a subject with which the author is exceptionally well acquainted, as the result of years of study in the field and of discriminating comparison of the views of other observers. A series of broad deductions are assembled on page 55.

A classification, illustrated by typical examples, forms the subject of Chapter VII. As stated by Mr. Weed, "this classification, though confessedly imperfect, seems to best express the character and relationships of the different types of copper deposits." Its main heads are: Igneous deposits, formed by segregations of molten magmas; emanation deposits, formed by vapors and gases from molten material; deposits formed by hot ascending waters; deposits formed by atmospheric waters; metamorphic and sedimentary deposits. Following this are notes, explaining and amplifying the classification.

In Part II. of the work are given descriptions, grouped by countries, of the principal copper deposits and mines of the world, which form the basis for the deductions presented in the first part. Though necessarily brief, in treating so huge a subject, the descriptions contain a large

amount of carefully arranged information. Many figures are given of size of ore-bodies, with sections and geological plans, analyses, value of ores and production. Among the most interesting sketches of mine maps and geological sections are those in the account of the Arizona copper districts. It might be thought that the author has carried brevity to an extreme, in allotting only eight or nine pages, including cuts, to the important Lake Superior copper region, but on comparing the descriptions of several prominent districts it is seen that a proper scale in apportioning the matter relating to them has been observed.

The book is well written and printed; it is valuable throughout, and will be read with profit by both consumer and producer of copper.

R. P.

Mine Timbering By WILBUR E. SANDERS, BERNARD MACDONALD, NORMAN W. PARLEE and others. New York, Hill Publishing Company, 1907. Pp. 177 (including index) and 140 illustrations. Price $2.00.

This book is composed of a number of articles which have appeared in the past four years in the *Mineral Industry*, *Proceedings of the Canadian Mining Institute*, *Transactions of the Canadian Society of Civil Engineers* and the *Engineering and Mining Journal*. Mr. Sanders contributes the leading article, of 50 pages, which deals generally with the timbering of shafts, stopes, drifts, etc., including cribbing, shaft-sets, stulls and square-set work, together with shaft stations, and timbering in connection with filling. At page 123 is another paper by Mr. Sanders, on the framing of rectangular shaft-sets.

The other articles deal specifically with practice in different districts or mines; thus, Mr. Parke writes of the methods of mining and timbering in large ore-bodies in British Columbia and Michigan; Mr. MacDonald, of mine timbering by the square-set system at Rossland, B. C.; Mr. C. St. G. Campbell, of timbering in Section 16 of the Lake Superior Mining Company, and Mr. J. H. Batcheller, of practice in the Cœur d'Alenes. The remaining articles, each only a few pages in length, by Messrs. L. S. Cates, C. T. Rice, W. R. Crane, T. J. Greenway and Mark Ireland, are on the system of timbering employed at Bingham, Utah, in the Lake Superior copper district, and in Queensland and Tasmania.

As stated by the editor in the preface, no attempt has been made to write a treatise on the subject. The different articles are left to speak for themselves. This procedure is attended by some disadvantages, among which is lack of unity. For example, Mr. Sanders treats of shaft-timbering in both of his articles, and the same subject appears in several other parts of the book, while at least three papers are on the square-set system.

Nevertheless, the book is a welcome one; the numerous illustrations are good and are given in much detail, often with complete dimensions, and the descriptive matter, sometimes comprising costs, is eminently practical in its nature. The only other book on the subject, of which the writer of this notice has any knowledge, is the short monograph by W. H. Storms, published in 1894 by the State Mining Bureau of Cali-

fornia. It is to be hoped that Mr. Sanders will follow his own suggestion, and publish a treatise on mine timbering in the near future. We understand that he has such a work in course of preparation.

R. P.

The Mineral Industry, its Statistics, Technology and Trade, during 1906. Vol. XV. Edited by WALTER RENTON INGALLS, editor of the Engineering and Mining Journal. New York and London, Hill Publishing Co. Pp. 954. Price $5.00.

The fifteenth volume of *The Mineral Industry*, larger in bulk than any of its predecessors, continues a work which grows in value with each successive issue. It not only constitutes a history of the industry, but enables the editor year by year to complete and correct figures of production which, when coming from remote districts, are sometimes difficult to secure with the promptness desirable in an annual review. The statistical tables, in which the summaries are grouped by countries, are given in nearly every case for a full decade, the last figures being corrected when necessary in the next volume issued.

The *Mineral Industry* has made a substantial place for itself in mining literature, because it presents in convenient and condensed form masses of information that otherwise would only be found scattered through the pages of the mining periodicals and transactions of engineering societies.

Besides the trade reviews and statistics of the different minerals and metals, written largely by specialists in their respective subjects, the present issue contains, as usual, a number of technical descriptions of methods of mining and the treatment and reduction of ores. The following papers may be noted: "Progress of Gold Milling in 1906," by Robert H. Richards and Charles E. Locke; "Fine Grinding and the Treatment of Slimes," by Alfred James; "The Cyanide Process in the United States and Mexico" (covering both gold and silver), by Charles H. Fulton; "Gold Dredging in 1906," by J. P. Hutchins; "Literature of Ore Deposits in 1906," by James F. Kemp; "Sampling and Assaying," by E. F. Colcord; "The Advances in Ore-Dressing in the Last Decade" and the "Progress in Ore-Dressing and Coal Washing in 1906," by Robert H. Richards and Charles E. Locke. The last named paper is a very full compilation of 55 pages. In addition to these special articles, much similar information is included in the other divisions of the work; for example, in the chapters on Lead, Zinc and Tin, by W. R. Ingalls; Petroleum, by H. C. George; Nickel and Cobalt, by John F. Glidden; Iron and Steel, by Frederick Hobart, with a review of the "Progress in the Metallurgy of Iron and Steel in 1906," by Bradley Stoughton. L. S. Austin contributes a long, illustrated paper on recent "Progress in the Metallurgy of Copper," and C. G. Atwater writes of "By-Product Coke Ovens."

Among other contributors to this volume are Edward K. Judd, F. W. Parsons, Robert W. Lesley, Geo. F. Kunz, R. Meeks and F. J. Tone.

R. P.

Nitro-Explosives. A practical treatise concerning the properties, manufacture and analysis, of nitrated substances, including the fulminates, smokeless powders and celluloid. By P. GERALD SANFORD, F.I.C., F.C.S., Public Analyst to the Borough of Penzance; late consulting chemist to the Cotton Powder Company, Ltd., etc., etc. Second edition, revised and enlarged, New York and London, D. Van Nostrand Company, 1906. Pp. xii + 300; 60 illustrations and diagrams. Price $4.00.

The first edition of this work was published in 1896. In the second edition considerable new matter has been inserted and recent developments in the manufacture of smokeless powders are treated as fully as the secretiveness of the manufacturers will at present permit.

An introductory chapter contains chiefly some general considerations relating to the location and design of explosives factories. Chapters II. to VI. deal with the properties of the different high explosives and their ingredients, including nitroglycerine, nitro-cellulose, or gun-cotton, the dynamites, the so-called flameless or safety-powders, the fulminates and smokeless powders. The remainder of the book, including Chapters VII. to IX., is on the analysis of explosives, their firing points, heating tests, etc., and the determination of their relative strengths.

The author has been practically connected, as chemist, with several explosives works and writes from that standpoint, rather than that of the user of explosives. Although the subject matter as a whole is perhaps of most interest to the naval and military engineer, still, the mining engineer will appreciate the information given as to the composition and nature of roburite, ammonite, bellite and other flameless powders, as well as the account of the advances made in recent years in the manufacture of smokeless powders, both in the United States and Europe. Unfortunately, these powders up to the present time have been too expensive to compete with the dynamites for mining purposes, but mining engineers will welcome the opportunity to use them whenever their cost shall have been sufficiently reduced. Their advantange would be great for shaft-sinking or for blasting in imperfectly ventilated underground workings.

The new edition of the work is valuable in giving in condensed form the procedure and results of recent practice in the manufacture and testing of the power of high explosives, but we think the author should have devoted more space to the effect on nitroglycerine of impurities in the ingredients, and to the methods of manipulation and the precautions taken to prevent heating during the operation of mixing the glycerine and acids. These matters are treated rather briefly, with perhaps insufficient comparison between the methods employed at well established works. The same may be said also of the description of the manufacture of gun-cotton.

In Chapter IV., on dynamite and gelatine, rather too much prominence is given to infusorial earth, or guhr, dynamite, and too little to the more important forms with explosive bases or dopes (page 119). The latter have in practice almost entirely superseded the former and yet the compositions and effects of the explosive bases are hardly mentioned and but little is said as to the question of the absorptive capacity

of different bases, the dangers of approaching saturation and the added power resulting from the employment of bases composed largely of explosive materials.

The book has many good features, however, and as a condensed manual is to be commended, but we think a fuller discussion of the points above mentioned should be given in a future edition.

R. P.

Boiler Waters. By WILLIAM WALLACE CHRISTIE. New York, D. Van Nostrand Company. 8vo, pp. 255, 77 illustrations. Price $3.00 net.

The object of this book is, to quote from the preface, "to furnish steam-users with information regarding water, its use, and troubles arising from the use of water, and remedies that may be used or applied."

The subject is treated in ten chapters covering: "Water, its Properties, Materials Found in Water, Water Analysis;" "Boiler Scale;" "Corrosion;" "Feed-water Pipes, Blow-off Pipes, Tubes;" "Priming and Foaming;" "Oil;" "Hardness of Water;" "Feed-water Heaters;" "Water Softening;" "Miscellaneous Tables."

The author has collected in this work, material of much value to the practical man in solving the problems which commonly arise in any steam-power plant. Much of the subject matter has been taken from journals not readily accessible, and the author is to be commended for having given references to the original in nearly all cases. Illustrations are numerous and exceptionally good. The chemical side of the subject is not exhaustively treated, but is sufficient for a clear understanding of the problems at hand.

The book can be fully recommended to all interested in the subject.

F. J. M.

Machine Design. By CHARLES H. BENJAMIN. New York, Henry Holt & Co. Cloth, 12mo, pp. 202. Price $2.00.

This work is an exposition of some of the factors involved in the design of some of the machine elements It is to be regretted that the author did not make his work more complete that it might really have been a work on the design of machines. While there are many works on the design of machine elements, there are none in this country which deal with the coördination of these elements in complete machines. Without this a work can hardly be called a treatise on machine design. The more important machine elements are considered in this text, and some of the principal factors involved in their design have been clearly set forth. A discussion on the properties of the materials of machines in Chapter I. leads to an understanding of the principal features involved in the selection of the materials required for machine parts. The student is supposed to have had a course in the strength of materials and to understand the formulas for the stresses in simple beams. Some of the most valuable features of the whole book are the results of experiments performed by the author on the strength of cast iron cylinders, fly wheels, gear wheels, pipe fittings, and bearings. A number of tables

constantly required by the designer are given, and many formulas for design of parts are conveniently grouped for ready reference. As an introduction to the study of machine elements the book is good, and the reader will be given a clear idea of some of the important features connected with the proportioning of machine parts.

W. R.

Chemical Reagents, Their Purity and Tests. A new and improved text based on and replacing the latest edition of KRAUCH'S "Die Prüfung der Chemischen Reagentien auf Reinheit." By E. MERCK. Authorized translation by HENRY SCHENCK. New York, D. Van Nostrand Company, 1907. 8vo, pp. vii + 250. Price $1.50 net.

This is an excellent translation of the acknowledged standard treatise on the purity and tests of chemical reagents, of which the Committee on Purity of Reagents of the American Chemical Society reported that until further work is done and more data collected any book which they might issue for the guidance of American chemists would be practically only a duplicate of the work of Krauch.

In the case of a work so well and favorably known it is only necessary to note the principal points of difference between this edition and its predecessors. Considerable condensation with a commendable reduction in price, but without loss of value in its own particular field, has been effected by omitting such descriptive matter as belongs more properly to general chemistry, as well as by the adoption of a somewhat more compact arrangement. The typography, paper and general make-up also show marked improvement over the English edition of 1902.

The present edition makes a most attractive and useful volume which should be in the hands of every user of chemical reagents.

H. C. S.

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Journal of Applied Science

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GEOLOGY.*

BY J. F. KEMP.

In the family of the sciences geology came of age at a comparatively late date. For a number of reasons this slow development was unavoidable. Geology could not mature without the aid of sister sciences which are necessary predecessors. The great, round world in its entirety cannot be grasped otherwise than with the assistance of physics, mechanics, astronomy, chemistry, zoölogy and botany. Geology is the application of all these to a particular planet, to the one which we count our home; to the only one of which we know by actual experience. Not alone was its early growth based upon the sister sciences but it now progresses with them, leans largely upon them for support, and in turn repays its debt by the contributions which it makes to each. Yet, after all, who will say that any branch of science stands alone, or that one is distinct from all the rest, when year by year the force of the old-time philosophical dictum "all is one" becomes the more apparent.

Geology, properly so-called, found little real development among the ancients and among the people of the Middle Ages. To be sure, the material world was all about the men of those times and they could not fail to observe it. Earthquakes shook the ground and volcanoes broke forth in eruption then as now. Miners were at work extracting the metallic ores from the depths and their

* Fifth lecture (November 13, 1907) in the Course in Science, Philosophy and Art, Columbia University, 1907–08. Copyright The Columbia University Press.

difficulties were not essentially different from ours. But the conceptions of systematic development in time, of effective change and of the long periods necessarily required were not originally, or at least not adequately grasped even by the speculative philosophers. In the general consciousness the mountains were the type figures of permanency and of unalterable resistance. Whereas to-day, as Tennyson expresses it, we think

> "The hills are shadows and they flow
> From form to form and nothing stands.
> They melt like mists, the solid lands,
> Like clouds they shape themselves and go."

This changed attitude toward the world of nature is one of the notable intellectual advances of the day. It has become involved with all lines of thought. From the pulpit we hear its conceptions cited. In the press the slow geological changes serve to illustrate an editorial point. And in the common speech of the people, some of its phrases have so ingrained themselves as to be almost second nature.

Broadly speaking, then, we may justly attribute to this branch of science an enlargement of our intellectual field of view, and at once a more dignified and a more elevated conception, not alone of the material universe, but of life itself.

In passing in review the problems, or at least some of them, upon which the workers of to-day are engaged, we may perhaps begin with those which bear upon the earth in its remotest stages of development, marking thus the connection of geology with astronomy and the mathematical sciences. We may then pass to those relating especially to later periods of time.

Faintly in the seventeenth century, and then firmly toward the close of the eighteenth, the general idea was gradually developed that the earth consists in its outer portion of concentric strata, which succeeded one another in time of formation. The existence of a very ancient core of hard crystalline rocks, called the primary or primitive, was demonstrated, and resting successively upon this, later ones were identified, which together built up what we summarize as the geological column. Not all of those familiar to us to-day were, however, recognized at the outset. Both at the earliest and at the latest extremes additional members have been interpolated.

Active and speculative minds were not content to think only of

the times which were actually represented in the solid strata themselves, but along lines partly geological, partly astronomical, partly also mathematical, they ran back to periods still earlier. Before the eighteenth century closed and quite universally throughout the nineteenth, the earth was believed by geologists to have once been a highly heated nebulous or gaseous mass and to have passed through stages of refrigeration to its present state. This conception is the one formulated in accordance with the well-known "nebular hypothesis" and it marks, as has been stated, the connection of geology with astronomy and celestial mechanics. Grave doubts have arisen in the minds of some, however, as to its truth and despite the significant astronomical evidence and analogies, the endeavor has recently been made by our fellow-countryman, Professor T. C. Chamberlin, of Chicago, to formulate an alternative conception, which would perhaps involve fewer difficulties. Instead of a highly heated, and subsequently cooled and solidified, gaseous original, minute particles of matter, which may have been molecules, are believed to have moved in orbits around a common center in a manner analogous to the solar system of to-day. In their evolution they became aggregated into larger bodies such as the planets and the earth, continuing in groups the motions and relations which they possessed when individuals. As the mass gradually increased, the pressure of the outer layers consolidated the core; and by the mechanical changes involved produced those internal stores of heat with which we are familiar in volcanoes and in deep borings and mines. Vapors or liquids in the original cold particles are believed to have been gradually squeezed out by this pressure. The little particles are called planetesimals or diminutive planets, and the hypothesis is styled the "Planetesimal Hypothesis."

It is perhaps too soon to forecast the influence of this new conception upon geologic thought. Like all attempts to formulate primeval conditions its data are partly matters of observation partly assumptions. Speculation enters in a very large degree, and, as in the case of the various and widely differing estimates of the age of the earth based on assumed rates of cooling, once the data are provided, mathematical reasoning goes to a conclusion with unerring accuracy. But the correctness of the solution turns on the reliability of the original data and where these are so largely assumptive the conclusions are from time to time subject to change.

The field geologist whose solutions of smaller problems turn upon carefully observed data, inevitably considers all the formulations of views regarding remote times and conditions essentially as working hypotheses, much less firmly established than many other results and to be taken much less seriously. Yet we must have a starting point and the striking contrasts of the older and the later views cannot but impress every one who reflects upon them. The former postulates a highly heated original; the latter a cold one. The one begins with gaseous matter; the other with solid. The one draws upon an original but diminishing store of heat; the other develops heat continuously by mechanical processes. In many ways the two are diametrically opposed; yet some have raised the question, whether, in order to obtain a swarm of separate cold particles, we must not in our thought go still farther back to a gaseous or nebulous source, and it is not clear that we have yet escaped the necessity of at least the essential feature of the nebular hypothesis.

But let us turn from these remotest and least tangible phenomena to the later ones of which we have the actual records. In all parts of the world, as we follow down through the successive layers of the fossiliferous strata, we find an ancient nucleus of crystalline rocks, gneisses, schists, marbles, serpentines, and minor varieties. The geologists who are now in middle life were quite universally taught by their instructors of twenty-five or more years ago, that the old strata were all sedimentary in origin, although now thoroughly recrystallized and of vast thickness. Not a few believed that they represented materials which had been worked over perhaps many times by the ocean and the other forces of erosion. The prevailing banded or parallel arrrangement of the constituent minerals was the trusted proof of their sedimentary character. But to-day when we look back we find that we have greatly advanced from these views and that we hold conceptions which much more satisfactorily fit the observations. We know that the banded arrangement of the minerals, light and dark, is not necessarily evidence of sedimentation, but that it may have been produced by a pressure, so great and irresistible as to cause the crystalline rocks to flow like viscous masses, and to drag and shear the minerals into long, parallel, leaf-like relations, suggesting indeed the bedding of sediments, but radically different from it.

While these conclusions have seemed inevitable to the observers

in the field, yet at least one inquiring mind could not rest content until the processes had been artificially reproduced in the laboratory. By a brilliant piece of work and with the aid of elaborate and expensive apparatus, Professor F. D. Adams, of McGill University has first tightly encircled cylinders of Carrara marble in jackets of tough sheet iron. He has then submitted them to slowly but irresistibly increasing pressure. As the strain passed their limit of crushing resistance, and while they were so tightly encased in the tough, but yielding jacket, that they could not crack, they have passed from cylinders an inch in height to discs, of half this altitude and of correspondingly increased diameter. Yet they have never lost their cohesion but have moulded themselves like so much wax. When removed from the jackets the discs are as solid and homogeneous as were the original cylinders.

These experiments are very recent, but Professor Adams is now continuing them upon other rocks such as granite; and instead of dry compression, now admits steam or heated water to the cylinder and seeks to reproduce in these respects the conditions of deep burial and metamorphism. In so far as the results have yet gone, they are a striking corroboration of the conclusions reached by observers in the field, among whom Professor Adams also occupies a most honorable place.

But we are likewise assured that banded or foliated structures may be original with igneous rocks. We have learned that molten masses in cooling and crystallizing sometimes separate into contrasted layers of light and dark colored minerals which may be brought by the conditions of eruptive flow into parallel relations suggestive of sedimentation but different from it. The field worker of to-day upon the oldest rocks therefore constantly keeps in mind the question, am I dealing with igneous or with sedimentary types? And the general result has been to prove in the areas of the oldest rocks a series of sediments indeed, but of greatly restricted amount and thickness as compared with earlier views, and to constantly abstract large fractional parts and prove them to be eruptive rocks, which have been forced through and between the ancient sediments. Vast masses of granite and similar types are now known, sometimes attaining dimensions of tens or even hundreds of square miles, which are later than the old limestones and sandstones, and which were regarded by our predecessors as one with them. On the whole the sedimentary antece-

dents of the fossiliferous strata have become much restricted in amount, and the time which they were once believed to represent has suffered diminution in a corresponding ratio.

A few moments ago the statement was made that at the close of the eighteenth century geologists well understood in a general way that the constituent strata of the earth could be classified in a systematic series of which some were older and some later. Soon after the middle of the last century the larger subdivisions were recognized, which we now call periods, and substantially the same names that we use to-day were given them. Geologists knew that before the appearance of life, or in the Archean times, there were the Laurentian and Huronian periods. During the era of ancient types of life, or the Paleozoic, there were six periods, respectively, the Cambrian, Lower and Upper Silurian, Devonian, Carboniferous and Permian. During the middle era, or Mesozoic, there were three, Triassic, Jurassic and Cretaceous. During the latest era, at whose close we ourselves are living, there were two, the Tertiary and Quaternary. These eleven periods since the advent of life were each distinguished by its own types of organisms, now fossilized, and each was believed to be quite definitely and sharply outlined. If a European or an American geologist were to visit Asia or Africa, it was expected that he would find the evidence of the same times, and of periods that were contemporaneous and equivalent with those already known to him at home. This conception lasted until well beyond the student days of geologists now living. But as time went by the point of view changed and one finds to-day a much modified attitude regarding this, the very foundation of the stratigraphic or historical portion of the subject. The old view was as if we should endeavor to coördinate all human history with the chronology of the Romans; whereas in China or India events were progressing without regard to Rome and or equal importance to the human race broadly considered. Multitudes of men, so great as to dwarf the numbers of the Mediterranean peoples by comparison, lived, moved, and had their being in Asia, quite unaware that a Roman had ever existed. So in geology, we have learned that while there is a general similarity of conditions over the world at any one time, yet the conditions have not been uniform and that each locality must be viewed by itself. We have all come to realize that so far from establishing contemporaneous life, identical fossils in separated localities indicate a

wide difference of dates. The studies of the paleontologist lead to conclusions similar to those of the zoölogist, that groups of animals or plants migrate from centers, spread abroad along belts of favorable conditions, sometimes driving out earlier inhabitants, and certainly for a time mingling with them until in the end they established colonies remote from their starting point. These migrations require long stretches of time, and radically change our old conceptions. The same species on opposite sides of the world means, therefore, not the same time for both, but a long interval between. Indeed, while in one locality organisms of a later type may be flourishing, elsewhere animals of earlier and more primitive characteristics may still be in the full tide of their activities. Just as to-day in Australia we have a fauna which is a survival of genera long since extinct in other parts of the world, so in the past, whose records have come down to us, the same contrasts have prevailed. What is Quaternary or Recent in one place, may be Mesozoic simultaneously in another.

You will see at once how much more complex is the history of the globe, when interpreted in the light of the higher criticism which places the observer face to face with actual conditions, and how changed is necessarily the attitude of the geologist of to-day regarding these fundamentals when compared with that of his predecessors, and even with the views of his own earlier instructors. Turning to the future outlook, shall we anticipate as have some the passing away of our established chronology? Shall we cease to speak of a Cambrian or a Carboniferous period as world wide in its application and of fairly sharp definition? Shall we reduce ourselves to local sections each with names of geographical derivation and suited to a limited area, only running comparisons in a very general way? Some geologists have been impressed with this as the inevitable development. But others, while appreciating the limitations placed upon the old divisions and the mistaken views which followed their precise and universal application, yet see in them when discreetly and wisely employed, the statement of a great truth. Realizing the indefiniteness of the boundaries, they yet cling to the underlying and vastly useful general application and will long continue to use the time-honored terms. The two periods of the Archean era, the six of the Paleozoic, the three of the Mesozoic and the two of the Cenozoic in not too sharp application are certain to be retained for many years to come, but

the tests of their identity must be more complex and must be applied with a more comprehensive grasp of the situation.

As regards the appearance and development of life upon the earth the two most interesting questions for the world at large, relate to its beginning and to its later stages. They concern on the one side the forms which the oldest living creatures assumed, and on the other the advent of man himself. We cannot contemplate these questions without having our thinking influenced by the evolutionary doctrines. The first remains of life in the Cambrian strata consist of hard parts such as the shells of mollusks, which were calcareous when the animals were alive and which retain in large degree their original composition. Only resistant and enduring materials of this sort could survive through long periods of time. They indicate highly organized creatures, and are generally believed to have been preceded by ancestors of simpler types. For the remains of these ancestors, geologists have searched the older strata with the care and persistence which the importance of the question demands, since to push the evidence of life back of the Cambrian is to win one of the great prizes in geological science. But hitherto the results have been most meager. They are limited to a few indefinite tracks or questionable impressions.

Opinion, therefore, has swung around to the conclusion that the earliest organisms were marine and that they were soft-bodied animals which on death left no trace in the sediments of their time. Along two lines of attack there is a most interesting corroboration. Professor R. A. Daly, now in Boston, has been led by detailed studies in the extended series of Cambrian and Pre-Cambrian strata along the national boundary between British Columbia and the northwestern states, to the conclusion that the Pre-Cambrian sea was devoid of dissolved salts of lime, and that, by reactions incident to the decay of soft-bodied animals upon its bottom it was robbed of such little lime as it might otherwise have had. For a number of seasons too complicated and detailed to present here, there is good ground for believing that not until the opening of the Cambrian was sufficient lime available in the sea water to encourage the growth of those hard calcareous parts which yield the fossil remains.

Along another line of attack very interesting conclusions have been reached. Dr. A. C. Lane, the State Geologist or Michigan, has sought to solve the problem of the composition of the oceanic

waters at various periods in the earth's history by a study of the deep artesian supplies which we gain from bore-holes reaching older and older beds. From the samples of course must be eliminated contributions from higher horizons of matter not contained in the waters at the time of entombment. When all precautions are exercised we discover in the end that the oldest waters are relatively dilute when compared with the sea of to-day and that they have but seven parts in the thousand as contrasted with the modern thirty-five. Common salt was the most abundant then as now. A very interesting connection of these researches with the problems of early life is furnished by a line of investigation pursued by Mr. R. Quinton, of the College of France. In all but the simplest organisms there is a so-called vital fluid which is the basis of the circulatory system. If, for instance, we eliminate from the blood of vertebrates or other animals the red and white corpuscles and the products of organic waste and then analyze the remaining fluid we find a weak solution of common salt in water varying slightly in the different types, but something like seven or eight parts in a thousand. This is believed by Mr. Quinton to represent the ancient sea water in which the ancestors of all modern organisms lived and from which some forms emerged in the past. Employed as a vital medium of circulation it became enclosed in those organisms which developed the interior sack having only communication with the outer world by osmosis, and its composition having been preserved, it recalls the early ocean, in which their ancestors originally lived. It is at least a striking coincidence that the composition of the vital medium is just about the composition of the sea as indicated by Dr. Lane for the early Cambrian or Pre-Cambrian times. If, as seems not unreasonable nor improbable, this composition, attained in the absence of salts of lime, was particularly favorable to organic life, then the Pre-Cambrian sea may have swarmed with soft-bodied organisms. And if, with increasing land areas, more lime poured in from their drainage and the composition of the sea changed, then some animals took to the land, others developed hard parts and left the fossil remains. But in the vital medium of their circulating systems they maintained a solution corresponding to the composition of the sea water which had proved most favorable to their nourishment in the previous marine period of existence. These suggestions are at least ingenious and turn our thoughts into new and promising channels which are worthy of careful exploration.

Toward the opposite extreme in the geological record of life upon the earth lies the problem of man's appearance, a question which links geology with anthropology.

It is only upon its geological aspects that I purpose to touch, and solely of two of its phases do I desire to speak. That the general attitude of educated people toward this problem has become more tolerant in fifty years is now well realized. One may refer to the evolutionary descent and antiquity of man without, as happened in the case of the famous Calaveras skull, thereby cutting short the life of a state geological survey. We were formerly too much inclined to mix up souls and bones.

It is now forty-eight years since Darwin's "Origin of Species" was first published and thirty-six since his "Descent of Man" appeared. The last-named work squarely propounded and defended an evolutionary development from ancestors among the primates. In anatomical relations to living types there was a profound gap between man and the highest apes, most marked to the general comprehension in the sizes and shapes of the skull and brain cavity. Interest became focused upon the hoped-for discovery in some hitherto unexplored part of the globe either of the still surviving "missing link" or else of fossil remains intermediate in type.

In the three or four decades which have elapsed since Darwin's books started investigations, no discovery of living man-like creatures of additional significance in these respects has been made; nor do we believe that such exist in the deep recesses of any unknown jungle. But in one notable instance the search for fossil remains has been more successful. On the banks of the Bengawan River, in Java, there are Pliocene or late Tertiary beds which for several years prior to 1895 were explored by Dr. Eugene Dubois, a military surgeon attached to one of the Dutch regiments. Dr. Dubois unearthed a partial skeleton of a man-like creature whose leg-bones showed that he walked erect and whose skull was intermediate between those of the highest apes and man. The volume of the brain cavity is two thirds that of the average man, and the outline of the skull is not very different from some of the lowest types already known among prehistoric remains. Suppose that we assume a critical attitude and insist that if man existed in these times, we question whether this specimen is a fair representative. We may consider it abnormal and below the average of his day. Nevertheless the presumption is strong that it is normal, since ab-

normal ones are the rare exceptions. Granted this, many have believed that the remains point to a separate line of descent from that of the apes, and that if we consider the fossil as indicating the ancestors of men, then the latter have come down a branch or phylum distinct from the line of the primates. We all await corroboration by more discoveries and generally anticipate that they will be made in the East, but we must admit that this Javan skeleton of *Pithecanthropus erectus*, goes a long way toward bridging the gap.

As to the presence of man in relation to the great ice sheet of the Glacial epoch, it seems now fairly well established that in Europe he was existant in the later stages. In America the evidence is less strong and abundant, and while such as we have has been subjected to much destructive, and, it would seem at times, strongly prejudiced criticism, yet there is some rather good ground for inferring that on this side of the Atlantic as well as in Europe, primitive man was living in its closing stages. The results, however, of patient and comprehensive investigation have operated to show the great complexity of the Ice Age. Its boulders and sands are of widely different and contrasted dates. We may not rashly assume its contemporary character in separated parts of the country. Man's presence in its closing stages but reveals the long duration which went before while yet portions of North America were in the grip of extreme cold. The best we have done is to show that presumably in the later stages man was existant.

I turn from the problems of life to that other branch of geology which deals with the architecture or structural features of the earth.

Their main types and varieties have been well described and classified for many years. The labors of the Swiss, French and German geologists in the wonderfully crumpled and faulted strata of the Alps and the no less fruitful observations of the Americans in the Appalachians have established the several kinds of folds and faults, the great structural features of the sedimentary rocks.

In the regions of extinct and dissected volcanoes, moreover, and in the localities of deep-seated eruptive rocks, the records of many patient workers have taught us of the forms and features of the igneous group. The excavations of the miner have revealed both sedimentary and eruptive rocks with an accuracy and minuteness of detail not afforded in any other way. To the general conceptions now firmly established in this branch of the subject we are not likely to add much that is new or important in kind, but within

the last few years we have seen them greatly change in degree. We have long known, for example, that excessive contraction of the crust first doubles strata together, then pushes the compressed fold over on its side, and finally forces the crumpled mass along above the beds lying beyond the folded area, leaving older ones stranded upon later. In the southern Appalachians, the overthrusts have carried the beds as much as five miles from their starting points and subsequent erosion has destroyed any visible connection. With some such distances as these we have long been familiar.

But recently the European geologists have worked out cases in the Alps, where the shove has displaced the beds as much as fifty miles from their old positions. It has therefore been necessary to enlarge our conceptions by a factor of five or ten. The discoveries are new and the distances involved are so great that some time will be required in which to readjust our old-time sense of proportions, but we all realize that in these greatly compressed areas the movement was propagated along a fault plane or a series of fault planes of flat inclination which began well down in the earth's crust and which probably once continued beyond their present lines of emergence. When, however, we seek some satisfactory cause or explanation of these phenomena, both smaller and greater, we are as yet not entirely successful. The contradictory riddle of every great mountain system still confronts us. For example, we know that from New York to Alabama that portion of the Appalachians which we ordinarily call the Alleghenies consists of sands, muds, shell-beds and coral reefs, the latter two often comminuted to slimes. In the case of the sands and the muds, the materials must have been worn by the waves from older lands, presumably to the eastward. They must have been laid down in shallow waters, layer on layer, until they became thirty or forty thousand feet thick. It is a difficult question to decide what caused the great trough to sink so gently that sedimentation nearly kept pace with subsidence throughout the long Paleozoic era. Why does the earth wrinkle downward in some localities for long periods of time and then, more remarkable still, why does it suddenly change its course and wrinkle upwards bringing the one time bottom of the sea into wave-like ridges which follow one another like so many great billows, petrified in mid-course as they roll in upon a shore. These evidences of past dis-

turbances even more significantly than the earthquake shock of to-day lead on irresistibly downward to the interior of the earth and to the problem of its physical condition, one of the most interesting with which we have to deal and one regarding which opinions have passed through many changes in late years.

When the geologists who are to-day in middle life were students, the earth was widely believed to consist of a crust, perhaps fifty miles thick, hardened upon cooling, and surrounding a highly heated, molten core which fed the volcanoes, which admitted of adjustments and which was the still hot representative of the old incandescent nebula from which all had been derived according to the nebular hypothesis. In my own student days, the attacks of the mathematicians upon this view were beginning to be felt by the geologists. The important consideration of the rigidity of the earth, resisting deformation as it does while rapidly rotating, the lack of internal tides and similar arguments were leading to the view that a vast liquid interior was an impossibility. The effects of the pressure produced by the weight of the outer portion which rests upon the inner core and its effect in raising the fusing point to temperatures impossible of attainment were also gradually realized. A compromise view became widely prevalent. It involved the outer crust which we knew from experience was solid, evidently to a depth measured in miles; an inner layer of fused material, intervening before the pressure became too great; and then a solid core. From the fused layer came the lava of the volcanoes. Its adjustments made the ups and downs of the mountainous regions possible and its shrinkage and that of its crust from loss of heat, furnished the contractile strains which primarily caused the folds and faults. The late Professor Joseph Le Conte, for so many years the greatly beloved instructor in the University of California, was a stout defender of this conception.

But the mathematicians returned to the attack and on the principles of sound mechanics maintained with force that nothing but an essentially solid and resistant globe would survive the evident strains to which it is subject. We must believe, they have told us, that from surface to center the earth behaves with no less rigidity than would a sphere of steel. Our honored colleague of recent years in this university, Dr. R. S. Woodward, has been one of the most forceful and influential of American scientists in upholding the contention. Gradually geologists have come around to this

view, but have modified it by a further conception which must be coupled with it in order to account in any satisfactory way for the evident phenomena of the mountains on the one side and the volcanoes on the other. The earth is solid, we admit; it is highly heated, certainly to very great depths; but all matter, however, brittle under the conditions prevailing at the surface becomes appreciably plastic when confined on all sides and subjected to pressures beyond its crushing resistance. Under these circumstances, rocks will flow as viscous media while losing in no respect their cohesion. In the deep-seated metamorphic rocks we see these evidences of flow, and that the flowage is possible, we positively know from the experiments of Professor Adams earlier cited.

We may, therefore, regard the interior as essentially solid, yet moving with a slow viscous flow in any direction of less pressure. It undoubtedly is highly heated, yet it is not a molten liquid as formerly regarded. If, however, during a viscous flow toward a point of less strain the interior rises toward the surface, it is subject to less and less pressure, and inevitably in time reaches conditions under which the rocks must fuse. The vapors and gases held in all deep-seated strata are freed, and the once solid rock becomes the molten lava with which we are familiar in volcanic eruptions. Volcanoes mark, therefore, at least in their inception, restricted areas in the upper world of such low or eased pressure that toward them the deep-seated rocks pass until they can flash into a molten and even explosive condition.

But there are some further interesting features which are displayed by the volcanic centers. We often find cases where practically the same vent has been active for a long period of time, yet where it has furnished successive varieties of lava in a more or less regular series. Rocks of intermediate composition are usually the first to appear. They yield on analysis fifty to sixty per cent. in silica and their specific gravity is of mean values. They are followed by others successively higher in silica and of lighter specific gravity. Then appears a marked drop to the last outbreaks of rocks, low in silica, high in iron and related bases and of relatively great specific gravity. Thus we have what Baron Ferdinand von Richthofen called many years ago "the natural succession of the volcanic rocks." Increasing experience has shown that we must admit some variation in the regularity of this series, and that there is sometimes a recurrence to the first variety, but on the whole, the

general succession holds good and involves an important truth. In order to account for it, we have come to believe in temporary reservoirs at depths not so great as to preclude fusion and in which the molten rock stands liquid for long periods of time. An original homogeneous mass then breaks up or differentiates into fractional parts. The lightest or most siliceous rises to the top, the heaviest or most basic sinks to the bottom. Successive eruptions are marked first by a rock of medium specific gravity and chemical composition, next by those lighter and lighter, or higher and higher, in silica as the upper layers are tapped off, finally closing with the heavy, basic dregs or settlings of the molten mass. The extinct volcanic areas of Nevada and Arizona have given fruitful evidence upon this point and where the original type of rock reappears after its normal period has passed, we have concluded that renewed viscous flow has fed again the reservoir from deeper seated stores.

The phenomena of volcanoes, the increasing temperature as we descend in mines or as we test deep bore-holes with self-registering thermometers as has been done most of all in America by our Professor Hallock, together with other less familiar phenomena which I do not cite, have all led us to the belief that the temperature of the earth continues to increase as depth grows greater. This conclusion is fundamental to some of the points just made, and is probably well justified. But as regards the profound depths the rate may not hold. Some mathematical students of the subject have reached the conclusion that the rate falls off greatly, and, regarding this, opinions may be said to differ, but there is no good reason to doubt that at least for relatively great distances the interior is highly heated.

Volcanic phenomena and the problem of the interior heat have been the objects of greatly increased interest in recent years because of the remarkable and destructive outbreaks in the lesser Antilles and from Vesuvius. In two particulars, Mont Pélè in Martinique has added phenomena previously unfamiliar. The descending blast of heated gas that wrecked St. Pierre with its tens of thousands of people was on a scale not formerly observed, and by its destruction of life aroused human sympathy as well as stimulated scientific research. The subsequent protrusion of the extraordinary spine of solid yet highly heated lava from the crater gave a striking illustration of a mass of rock passing upward in a

viscous state, then solidifying before it emerged, but remaining red hot and steaming after it projected above the surface. Unfortunately it has fallen away and no longer stands as a monument to the dead buried at the foot of the cone.

There is one very recent suggestion regarding the problem of internal heat, with its corollaries, the volcanoes and the earthquake. It is the possible bearing of radio-activity. The recent discoveries that certain rare elements or so-called elements pass through a series of changes, ordinarily described as "breaking down" into other substances, with the parallel development of heat; and the further discovery of the radio-active character of the volcanic rocks and their emissions, have brought prominently before our minds the possibility that herein lies a cause of the internal heat and of the emitted lavas. The idea is so new that it is as yet little more than a suggestion, but we all realize that it may be fraught with important consequences for the future and its development in the years lying immediately before us will be watched with intense interest.

It has been impossible to pass in review the topics hitherto treated without constantly introducing the idea of time, and of time so indefinite in character that we describe it as geological in distinction from the usual measurable durations. Geology brings us irresistibly face to face with stretches inconceivably long which elude our grasp as do the distances of the astronomers. Some bold mathematical spirits have sought to calculate the age of the earth from the assumed nebulous conditions to the present. The results have varied from twenty millions to several hundreds of millions of years. Perhaps one hundred millions is generally accepted as a fair attempt at a solution upon such uncertain data. If we drift away from the nebular hypothesis we lose even these data and must attack the problem along new lines. If Dr. A. C. Lane succeeds in establishing from his samples of deep well water, the composition of the Precambrian sea, and if we can calculate the rate at which rivers have poured their dissolved salts into the ocean then we may attain an approximate idea of the time period required to raise the ocean to its present concentration.

With all these very long durations, great uncertainty is involved. Better success has been met with smaller ones, and in several cases the solutions attain much interest. The Falls of Niagara have furnished on the whole the most significant instance, and the one upon

which the greatest efforts have been expended. We can but wonder if when Father Hennepin looked upon them in 1768 and felt overwhelmed, as he tells us, with the tumult of the waters, the thought did not pass through his mind as to the length of time during which the river had been at work since the crest was seven miles away toward the north. The question looks so simple at first glance that it seems as if it were merely a matter of arithmetic to observe for a time the rate per year of the recession and then divide this into the length of the gorge. If the recession is two feet per year, and the gorge is seven miles long, something like 18,000 years would be required. But we have learned that there are a number of uncertain factors. While the American Falls may retreat less than a foot per year, and even for ten years at a stretch draw back less than two inches annually, the Canadian Falls after ranging about two feet for many years may average five for five years running. The crest is now very broad; yet obviously it has been much narrower throughout the greater part of its history. At the Whirlpool Rapids we find an older channel, one antedating the Glacial epoch, which must have headed once somewhere up stream and which can be followed out to the north. Thus, the Niagara River, as we know it on the retiring of the great ice sheet, found a part of its gorge already cut. The part was probably a modest fraction, but we have no means of determining exactly how much. It has also been demonstrated that below the Whirlpool Rapids, at what is called the Foster Flats, the river ran in a narrow western and broad eastern channel with a small island between, much as now, but on a different scale. Still making all allowances for these facts and variations, and recognizing that the present Niagara began its work when the continental glacier in retreating uncovered its drainage area and outlet, we generally believe that not less than 10,000 nor more than 50,000 years have elapsed.

Within the last few years geologists have been impressed with the line of attack furnished by measurements of radio-activity. It has been shown that not only eruptive rocks, but also both the waters and the surface deposits of up-rising, deep-seated springs display these phenomena in determinable degrees. In the latter case the radio-activity of freshly-deposited material is the greatest, and from this condition the amount falls off at a fairly well understood rate per year. If an observer, for instance, in the Yellowstone Park determines the radio-activity of freshly-deposited calca-

reous matter at the Mammoth Hot Springs, and then makes a similar determination on ancient incrustations at the base of an abandoned terrace, the lapse of time since the latter began is within his grasp. In this way we have learned that a thousand years ago a certain deposit began on the glacial drift, and with more determinations of a similar character we may learn when these hot springs first began. The method is very new, but promises results both of importance and of general interest. So new is it that at the moment we are better able to mention its possibilities than to give results *in extenso.*

Geology touches human affairs most closely on the side of mining and related enterprises, and there is no more interesting chapter in its history than the one relating to these services. Mining is a very old occupation. Civilization could scarcely advance without the aid of the metals and modern civilization owes one of its greatest debts to coal, the common source of power.

When, however, an engineer is face to face with the actual problems of removing these materials from the earth in the most economical way, and when he must plan such operations for a period of years, it is fundamentally necessary that he should have correct geological conceptions of their origin and structure. That our usual coal seams are ancient accumulations of vegetable tissue, deposited under quiet waters in the long course of swamp growth, no properly informed person can longer doubt. That they have then been folded, often faulted, and always eroded until we have but the remnants of their past extent, admits of no question. The simple facts and principles of structural geology lie at the foundation of this branch of the mining industry and have been long established. Nevertheless, as to the exact processes which have brought carboniferous or cretaceous muck, peat and lignite along the road to the distant goal of anthracite, we are yet in debate. That the original plant tissue of the anthracite seams was so long exposed before burial under the sediments which followed hard upon it, that it had lost the greater part of its volatile constituents and readily became the dense, hard, smokeless coal which we in the East so much prize, is the acute suggestion of Professor Stevenson, of our sister university here in New York, and his explanation fits the Pennsylvania case much better than any other which has been made. Yet not a few are pondering the question and some think that a more open-textured cover of rocks has permitted the gases

incident and necessary to the change to more freely form and escape in the anthracite areas than in the bituminous. Some again think that the mechanical heat produced in folding, or the interior heat of the earth made available by relatively deep burial under later strata, or the neighborhood of lower lying but as yet undiscovered eruptive rocks with their stores of heat, have brought about the changes. Upon this problem many are therefore still disposed to ponder.

Again, speaking of coals both bituminous and anthracitic, we may question as follows: Were they once leaves, twigs, branches and woody tissues generally, or were they largely spores of cryptogams. If we also admit to our field of view the interesting group which embraces the cannels and the exceptionally gaseous coals, from which to this day illuminating oils are distilled in Scotland and in New South Wales, we may ask: Were they enriched with animal matter such as fish, making them fry and sputter in our grates like so much fat; or were they due to the enormously rapid multiplication of fresh water algæ in a single season, as Professor Bertrand of Lille, has inferred from his microscopic studies of the Franco-Belgian occurrences. If this is true, then for these coals our old ideas of slow accumulation and protracted time are unnecessary and carbonaceous deposits may have gathered with a rapidity of which we have had no proper grasp. And then there are the strange asphaltic coals which fill the fissures across our strata of sandstone and shale; representing thus the inflow of long extinct petroleum springs, now evaporated and oxidized to a shining black residuum — vein fillings they are, analogous to those metalliferous deposits to which I will shortly pass. They connect the coals with petroleum and natural gases and lead us to ask: Whence came the oils that gave rise to them and to our subterranean pools which have so long remained stored up in the liquid form awaiting the drill? Are they, too, fractional products from long-buried plants and animals, gradually emitted during decomposition or gentle distillation, as almost all of us in America believe; or are they outgivings from molten rock which has never reached the surface and which has been forced to steam and fume in the confinement of the lower world? A few observers urge this view and a still smaller number, mostly laboratory workers, without the steadying influence of field experience, have suggested that there are inner zones of the earth composed of metallic carbides, or even of

metallic alkalies to which waters charged with carbonic acid and atmospheric air penetrate and are there changed into petroleum and natural gas. These latter are views flatly contradicted and rendered impossible by the assured physical condition of the interior.

From asphaltic coals in fissures and veins, we pass to the metalliferous minerals in similar relationships. Always when we speak of veins, the precious metals more especially fill our field of view, and what romantic and fascinating chapters in human history they suggest! We mining people think that from the time when primitive man realized his need of metals, no influence in the world's history has been so potent to cause migrations as the desire to find the silver and the gold of new countries. The Spaniards certainly came to North, Central and South America chiefly to gain them. The Englishmen hastened in droves to Australia. Occidental and Oriental alike formed the mad rush to California. The stolid Dutchman of South Africa was swamped by the flood of English, German, French and American gold-seekers on the Rand. Nor in our own day could the icy barrier of Alaskan cold stave off the tide of fortune seekers. We sometimes think that in the settlement of this staid and sober part of America the longing for religious or political freedom almost solely brought the settlers. But the legends of the old silver mines still linger in the Highlands of the Hudson, and the hills of New England do not lack shafts and pits now long gone to ruin. Who will say to what extent these monuments to dead and buried hopes may not indicate the alluring influences of gold and silver in the unknown wilderness.

Ibsen has well said in his little poem, "The Miner":

Break in thunder, wall of rock,
At my hammer's tempest shock.
Myriad voices of the mine
Call me to its inmost shrine.

Glistening spirits beckon me,
To their sunless treasury,
Veined gold, all burning bright,
Diamond and chrysolite.

What though darkness be my lot,
Strike my hammer, falter not,
What though every hope be vain,
Strike my hammer, strike amain.

Wherever rich enough to mine, the metals are found in a greatly

concentrated state as compared with the earth at large. They so often fill crevices which traverse the rocks as veins cross the human hand, that the name vein has long been applied in this connection. The figurative use of the word is also happy, in that the metallic minerals, as well as their associates of no value, have been obviously brought to their resting places by circulating waters, which in the miner's phrase have nourished and enriched the crevices as does the blood the body. The veins again are, as it were, wounds or gashes in the earth which have been healed by the circulations. That water has accomplished the task few doubt. The earlier observers were accustomed to attribute the subterranean waters to the rainfall, and, until recently, all of us who have specially studied these phenomena have fallen in with the same view. From the clouds the rains descend upon the earth's surface. In part they run off in the stream; in part they evaporate, and in part they pass into the ground, where for ages they have accumulated as the ground-water and where they constantly renew the supply. The ground-water fills our wells, pours into the shafts of our mines as we sink and forces us to pump. It stands beneath almost all parts of the surface, at depths depending partly on the amount of rainfall and partly on the nature of the rock. We formerly believed that the ground-water continued downward in the crevices of the rocks to great depths, far enough in fact to come under the influence of the internal heat of the earth, and that it was set in circulation partly by this and partly by the heat afforded by those portions which, entering the ground on the heights, forced up the shorter columns after a greater or less underground journey. In the depths where heat and pressure were influential the waters were believed to be active solvents, and in their wanderings through the narrow pores and spaces of the rocks they gathered up the metallic and non-metallic minerals and deposited them in the veins when they rose to higher levels. In the journey toward the upper world diminishing heat and pressure led to precipitation of dissolved contents. In former years we had no very definite conceptions of accessible depths, but the mathematical analysis of Professor Hoskins, of the University of Wisconsin, combined with the geological observations and reasoning of President Van Hise, of the same institution, has shown that cavities become impossibilities in the softer rocks at say 500 meters, and in the most resistant at or about 10,000. That is to say, under tbe irresistible crush of the overlying column,

rocks hard or soft weld together into a solid mass. Circulating waters descending from the surface must be limited to these depths as the extreme.

But in the shafts, which are rapidly exploring lower and lower rocks in many localities and are changing inferences to observation, the experience of the miner has shown that in regions remote from recently extinct volcanic activity, the ground-waters end at a comparatively shallow distance. If while the shaft goes constantly downward the miner impounds the waters encountered at the upper levels, he meets no more below a thousand or fifteen hundred feet. But the veins are known in many cases to continue certainly to three times this depth, and we have been forced to cast about for some other source than the rainfall for the subterranean circulations which have originally brought in the ores. Other waters have been suggested by two chief facts of observation. Whenever the volcanoes break forth in eruption, clouds and clouds of steam and other gases go violently into the atmosphere, and when the lava wells upward from the vent and pours abroad over the crater slopes and surrounding country, it steams for many months before it is exhausted of its dissolved vapors. Molten rock in the depths of the earth must be well nigh saturated with the pent-up gases, which are only held in check by the surrounding envelope of the crust. We have even thought, with much reason, that the steam given forth by eruptive rocks in early stages of the earth's history has condensed as the oceans, and that thus the oceans have been derived from the volcanoes, not the volcanoes caused by the percolating oceanic waters according to the more popular view. The vapors of raw materials for subterranean waters are therefore certainly in the reservoirs of interior molten rocks.

The second significant point is that in almost all of our own mining districts where the operations are based on veins, eruptive rocks are a marked feature. In many there seems no escape from the conclusion that they are in some way intimately associated with the formation of the veins which have come in as the end-product in their expiring activity.

Thus a very large proportion of mining geologists and an even larger number of mining men accustomed to deal with the familiar phenomena of western ore-deposits, have swung around to a belief in the waters from the eruptive rocks as the agents and vehicles of deposition. Even the prospector looks upon the eruptive dike

as one of the best guides in his searching. When, therefore, a mass of molten rock charged with its burden of dissolved vapors and gases rises from the depths, and stops without reaching the surface, it slowly cools, congeals and emits the vapors as the source of the mineral bearing waters which continue onward to the upper world. They start highly heated, under great pressure. The *vis a tergo* is enormous. They constantly cool as they go and the pressure diminishes. The dissolved minerals precipitate upon the walls of the fissures, quartz and gold, in one region, silver ores with copper, lead or zinc in another. Layer on layer they grow, until the water course is choked or the fires below burn out, the steam pressure falls, and the circulation ceases. Water on the surface now begins its work, the rocks are worn away, the fragments of the vein roll down the mountain side and form the trail, which catching the eye of the prospector, guides him to the outcrop.

In formulating this brief statement I am not unmindful of the uncertainties involved nor that we have launched out from the facts and realms of actual observation into the profounder depths which we cannot attain. And yet by growing experience these views are found to fit the case in a satisfactory way and our philosophical ponderings on the subject are consequently fraught with fewer difficulties and contradictions than before.

There have been advances in other branches of the subject upon which I may not dwell in the short space of an hour. Geology applied to geography under the now current term of physiography has given old facts new meaning; has made the dry bones assemble again to a living organism. Political boundaries, the distribution of races, often the turning points of history are conditioned by some mountain barrier or are due to some plain or river. The relief of the surface, the upheaval of the mountains, and the erosion of the valleys are now recognized as powerful factors in the history of mankind. The patient labors of many geologists for a century and a half in recording and studying their causes and describing their characters have borne fruit of unexpected richness and quality.

But has geology made no broad and general contributions to human thought and life? Surely when we pause for a moment and reflect upon the changes through which our views of the earth and its development have passed in the century and a half now closing, we must admit that its influence has been enlarging and

beneficial. Recall if you will the gross ignorance that prevailed about the age of the earth ; the ridiculouly insufficient conceptions of time ; the idea that fossils were the sports of nature, the *lusus naturæ*, entombed in the rocks to deceive or amuse mankind; the failure to grasp the plainest lessons of geological structure; and the absurd positive notions of the earth which once prevailed. Then we realize that into the mind of every man of even ordinary education and reading has come a flood of new light. The earth is viewed not alone in a way that is more correct and nearer the truth but more dignified and stimulating.

When, in the striking and suggestive lecture with which this course opened and which made us all sit up and think, Professor Keyser described the inner world of the spirit to which the mathematically inclined may retreat, he awoke a vibrant chord in many minds. Doubtless there were others besides the present speaker who harked back in their memories to old student days with the higher mathematics, when in the Elysian fields of pure thought, oblivious to space and time, comforted perhaps in affliction, absorbed and indifferent even to pain, they roamed in association with the great minds of the past. It seemed like a glimpse into the golden age or better still a vista into those future realms of the spirit toward which we now and then turn our longing eyes.

Has geology nothing similar to offer? I think it has although in a different way. Not to the realms of the inner spirit may we turn, but abroad, comprehensively, over the face of nature. Under the guidance of this branch of science we see the long and stately panorama of the past unfold. Mountains rise and disappear; sea and land repeatedly change places. Majestic forces in orderly succession perform their work ; and development, whose course we may follow and whose processes understand, is made clear before our eyes. We feel ourselves as it were upon some mountain summit looking ever away and away into regions beyond. And just as when standing amid such surroundings we have felt the ecstatic thrill brought by the expanding prospect and have burned with eagerness to penetrate the new lands farther on ; so in the realm of geological thought we look back over the periods of the past, over the course which we have come, over the problems solved, the lower summits climbed, and then turning ahead to the future, to the yet unknown, we burn with eagerness to push forward in our day and generation the boundaries of knowledge another stage.

QUALITY OF BLUESTONE IN THE VICINITY OF THE ASHOKAN DAM.

A Study in Applied Geology.

By CHARLES P. BERKEY.

A great dam is to be constructed across Esopus Creek in the southern Catskills of New York to hold the waters of a reservoir covering about 9,000 acres, that is expected to furnish New York City with 500,000,000 gallons of water per day. It will stand 220 feet high from the rock floor of the creek valley and in its construction nearly a million cubic yards of material will be used. The greater proportion of this material is to be stone used as concrete filling, cyclopean masonry and dimension stone for facing. The quantities involved make it especially desirable that abundance of stone should be found in the immediate vicinity; but of even greater importance is the assurance that the quality is all the rigid requirements of such an extensive structure demands.

All of the more prominent outcrops of rock ledges and quarries, looked upon locally as available for this work by reason of location or reputation, were visited.

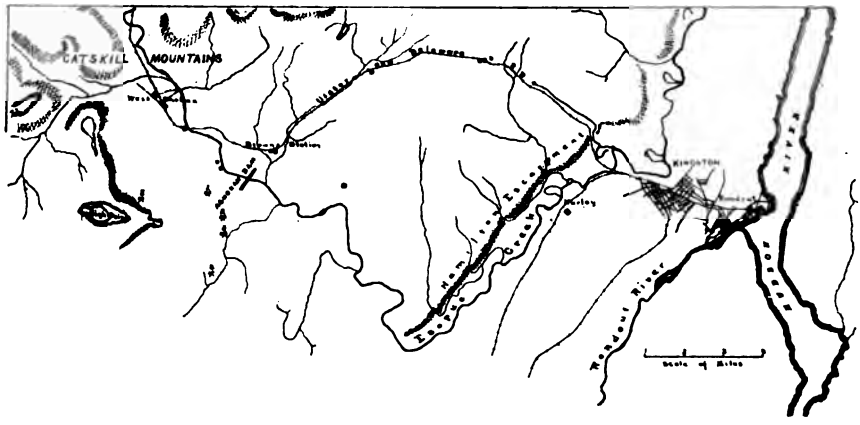
The following discussion summarizes the observations and arguments and conclusions of the writer in a study of this problem * in applied geology.

Part I. Geology.

Physiographic Features. — All of the rock formations of the region are sedimentary, chiefly sandstones and shales. They lie in alternating beds of variable thickness and are almost horizontal. The total thickness is many hundred feet so that neither the bottom nor the top beds of the series are to be seen in this locality.

The region is one of considerable relief representing Pre-Glacial erosion. The glacial drift mantle has modified it chiefly by obscuring some of the smaller irregularities of rock contour, and

* This study was made originally as a preliminary report to J. Waldo Smith, the chief engineer of the Board of Water Supply of the City of New York. The form has been changed but the facts and arguments are the same as in the original report.



especially by partially filling many of the stream gorges. Post-glacial erosion has not completely reëxcavated the old channels. But the contour of the uplands reflects the character of the bed rock with considerable success. The tendency of the more massive and coarse grained varieties of rock to resist weathering and erosion more successfully than the finer grained and more argillaceous or shaley facies is a general characteristic. Since these varieties form successive or alternating beds throughout the whole area, the result is an almost universal cliff-and-slope surface form. This bed rock topography is somewhat obscured but not wholly obliterated by glacial erosion and deposition. Therefore it may be used with confidence in locating or tracing the more durable beds since they almost invariably appear as a shelf or terrace with a steep margin toward the lower side and a gentle slope on the rising side.

Structural Features. — The rock types include bluish-gray or greenish-gray sandstones with almost horizontal bedding, and sometimes exhibiting cross-bedding structure, and compact very dark argillaceous shales. These two are of about equal prominence, but only the sandstone is of importance in the present discussion. Its minute structure will be discussed in greater detail in the petrographic part of this article.

Jointing is common and persists in two sets nearly at right angles to each other — one striking northeastward and the other toward the northwest. In some of the best exposures, these joints are clear-cut and run 10 to 18 feet apart, dipping almost vertically. In the more massive beds there is very little small jointing, so that the character is especially favorable to large dimension work.

But still more prominent structures are the partings which follow the bedding planes. These give the rock a decided tendency to cleave naturally into slabs, the uppermost exposed portion of almost every outcrop exhibiting this slab structure in more or less perfection. So general is this structure at all horizons in the sandstones of the series that there can be no doubt of its connection with some original sedimentation character. Besides it is a potential factor in nearly all the beds even when not very apparent. The exposed places exhibit the character so prominently only because of the weathering effect, which only develops the natural tendency. This general conclusion is borne out by the well-known practice of quarrymen of the district of splitting the larger blocks into slabs of the required thinness by wedges driven along certain streaks that are known as "*reeds*." A reeding quarry is one that has this capacity well developed, and it is this character in part that has made the "bluestone" or "flagstone" of New York an important factor in the production of the United States for a great many years.

For large size dimension stone where great stress is involved it is evident that this structure would not be desirable. These definite planes of weakness reduce the general efficiency. A little observation however shows that there are some outcrops and an occasional quarry where the more massive blocks do not split well. From the necessities of the industry these have been avoided or but meagerly developed. In some cases of this kind the sedimentation is of the cross-bedded type with somewhat interlocked laminæ. If the grain is coarse such varieties resist splitting with great success.

The thickness of such beds varies from a few feet to 25 feet or even more without prominent interbedding of shale layers. The record of a 100-foot drill hole, put down in Esopus Creek, near the site of the proposed dam, will illustrate the usual changes of character.

Top.

16 ft. coarse sandstone.
1 " conglomerate.
1 " sandstone with shale streak.
4 " conglomerate.
5 " fine sandstone, somewhat variable.
11 " fairly coarse sandstone.

8 " fine argillaceous shale.
2 " sandstone.
2 " sandy shale.
2 " fine sandstone.
8 " coarse sandstone with occasional streak.
35 " compact shale.
Bottom.

Stratigraphy. — These are the sandstones, flags and shales known as the Hamilton, Sherburne and Oneonta formations belonging to the Devonic period. The strata of the immediate vicinity of this examination probably belong to the Sherburne subdivision, but no attempt to differentiate the formations was made Structurally and petrographically the different formations are not distinguishable in this area. On the market the stone from either is known generally as " Hamilton Flag " or " Bluestone."

Economic Features. — There are hundreds of quarries in this general region. Nearly all are small, and are worked on a small scale without machinery. The product is almost wholly thin slabs of the flagstone type. This is supplemented by a small amount of somewhat more massive character, dressed for window-sills; and a very limited output is of dimension stone of larger size. The general lack of suitable mechanical devices and transportation facilities are the chief reasons for the limited output of the last-named grades.

Character and Conditions most Desirable. — In view of the particular use that is proposed in this case, it is desirable that the following conditions should be met:

1. The rock ledge should have a good attitude — *i. e.*, it should lie in such way that a quarry may be opened on it with small amount of dead work in removing drift cover or worthless rock.

2. The proposed quarry must have a good location — *i. e.*, it should be within two or three miles of the dam-site, and be above the 600-foot contour line.

3. The ledge must be continuous over considerable area. There must be opportunity to open extensive quarries for large quantities of stone. This applies both laterally and in depth.

4. The stone must be of good quality. It must be free from shale layers. It must not exhibit strong slab development. It must not contain a prominent reeding tendency. The larger blocks that are obtainable the better. Those quarries commonly

avoided because of their poor slabbing capacity would appear to be the most promising for this particular purpose. Especially the structure indicating an interlocking of laminæ, or enough such irregularity to obliterate the so-called "reed," is absolutely essential. If the proper quality can be found, neither the weather nor the stresses of the dam will be likely to develop special lines of weakness.

Recommendations. — Most of the quarries examined are unsuitable either as to type of rock or location. Four, however, that have acceptable location and attitude, are believed to exhibit the features and qualities that are most essential with sufficient prominence to encourage closer investigation as to extent and quality. These four quarries produce good qualities of stone that can be obtained in dimension sizes. None of them is very extensively worked for such quality and therefore the extent is undetermined. But it is believed that in two there are sufficient quantities.

These four quarries together with one other on High Point Mt. are arranged in order of relative distance from the dam-site.

1. The Yale quarry about a half mile southwest of the dam-site is the nearest. It is very favorably situated both as to distance and economical development. The rock is extremely massive with cross-bedding tendency. Considering its location and the possibilities of extensive quarries and its apparent quality this is probably the most promising ledge to explore. Its limited development prevents as definite observations as can be made at the other quarries to be enumerated, but there is every surface evidence of a thick bed of first class stone.

2. The Davis quarry is a little over a mile distant. The quarry has a rather strong "reed." But the lowest beds are massive.

3. The Keator quarry is about a mile and a half south of the proposed dam. It is not much developed and the extent of the desired quality is uncertain.

4. The Beesmer quarry is about two and one half miles south of the dam-site. This ledge is strong, and first class rock and probably extensive enough. Other quarries located along the same bed may be of the same quality. It is believed to be, from the petrographic standpoint, the strongest rock examined.

5. The Boice quarry on the northern slope of High Point Mt. has excellent quality of rock. But in other respects it is not so encouraging. Its attitude and the great amount of less massive

rock cover and the distance from the dam makes its use unlikely if any of the others prove to be suitable.

Both the Yale * and the Beesmer ledges will well repay further exploration by drill boring and other investigation, and are as good quality as the local rock is likely to afford. The chief question to be determined is the extent of the first grade rock. Extensions of these same ledges may prove equally encouraging much beyond the present limits of the quarries named.

These general observations indicate that suitable rock for use in the Ashokan Dam for rubble, concrete, cyclopean masonry, or dimension stone is available and near at hand.

Part II. Petrography.

The basis of this discussion is a microscopic examination of several thin sections made of the different types of rock from the quarries whose field geologic features, as outlined in Part I., give promise of encouraging results. The most characteristic variations are illustrated in the accompanying photomicrographs, Figs. 1, 2, 3 and 4.

Texture. — The rock is granular, the individual grains varying from minute particles in the finer shale layers to three or four tenths of a millimeter in diameter in the coarser sandstones (Fig. 2). The grains are seldom rounded. Jagged or frayed or elongate forms are the rule (Fig. 1). There is no marked porosity. When the rock was first deposited as a sediment it probably had the usual large interstitial spaces of such rock type, but in this case some subsequent modification — an incipient metamorphism — has obliterated the voids by the introduction or development of mineral matter of secondary origin.

In general it is quite apparent that the average grain was originally more rounded than its present representative.

Mineralogy. — The original minerals in order of abundance were the feldspars, quartz, and probably hornblende, biotite, and in much

* One of the diamond drill holes subsequently put down at this place to test the extent of the ledge penetrated to a depth of 68 feet. The records show 7 feet of clay and sand, 57 feet of first and second grade sandstone and 4 feet of shale and poor grade rock. Another shows 4 feet of clay and sand, 45 feet of first and second grade sandstone and 11 feet of shale and poor grade rock. Numerous other holes have been drilled, on this and adjacent ground, with results that indicate comparatively rapid changes in quality in the same beds within very moderate distances, and showing greater variability than is usually credited to these formations.

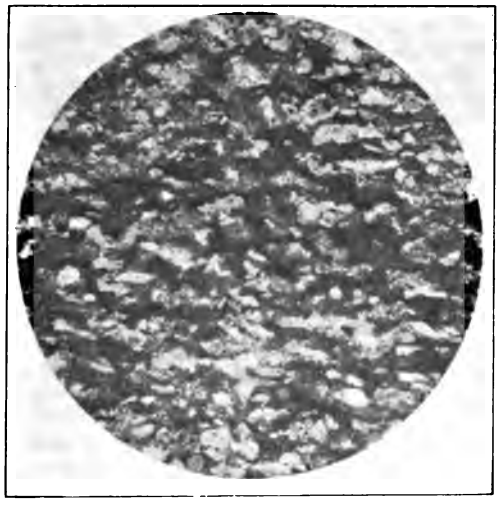


FIG. 1. Photomicrograph of reeding quality bluestone, — × 25 diameters. Intended to show small size of grain, their elongation, and the tendency to parallelism.

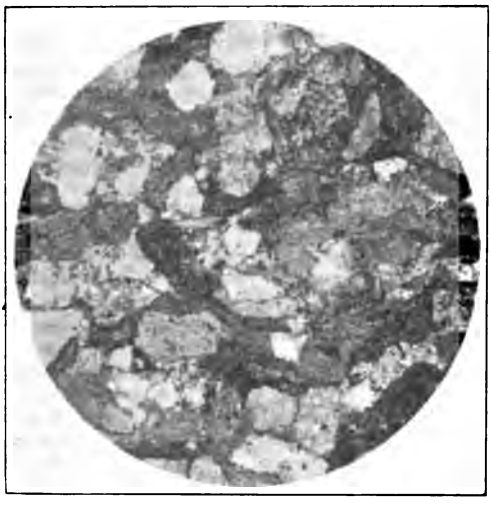


FIG. 2. Photomicrograph of coarse-grained bluestone of the first grade, — × 25 diameters. Shows typical association and distribution of grains in the most massive varieties. The clearest grains are quartz, the others are chiefly feldspars modified from their original condition. The tendency to fibrous development and the closely bound structure are noticeable.

smaller amounts others of little apparent consequence in the present discussion.

All of these have been more or less affected by subsequent changes. Quartz has suffered least of all, the chief modification being a greater angularity of form and an occasional interlocking tendency caused by secondary growth (Fig. 3).

Both orthoclase and plagioclase feldspars occur. The orthoclase grains, which originally made up more than half of the bulk of the coarser types of rock, has been in places profoundly altered (Fig. 4). In many cases the identification of this mineral depends upon its association and the abundant remnants of characteristic structure and its normal secondary products. In the least affected grains satisfactory identification is not difficult. Even in the most modified representatives there is some preservation of structure indicating size of grain and proving the essentially granular character of the rock. The plagioclase, although not abundant, is more readily detected than the orthoclase because it has been much less affected by the secondary changes.

All original ferro-magnesian constituents are wholly altered. There were some such constituents in the rock, as is plainly shown by the secondary products. Hornblende and biotite were probably both present.

The secondary products, derived from the original feldspars and ferro-magnesian constituents, include sericite, chlorite, calcite and quartz as the most important and abundant. Others probably occur that are less readily differentiated, and among them is kaolin. Occasionally a small amount of massive or granular pyrite occurs. There are traces of organic remains, especially plant stems, and the pyrite is most plentiful in association with those beds.

It seems to be the secondary products largely that give the characteristic bluish or greenish color to this stone. Practically all of the iron freed by secondary changes from the ferro-magnesian constituents has entered into new silicate compounds, especially with the chlorite, which are minutely distributed throughout the whole mass, giving it all a tinge of the characteristic color of those well-known products. The same amount of iron in the oxide form would no doubt give as highly colored stone as any of the reds or browns of other familiar types of sandstone. But the tendency to form the sericite-chlorite-quartz aggregate in this rock has also an

important bearing on its durability and strength. This is further discussed in a separate paragraph.

Classification. — It is clear that this type of bluestone is a sedimentary rock of medium grain, a sand-rock or "renyte." Since the silicates are so predominant in the original composition it may be further identified as a sandstone or a "silicarenyte." But in view of the predominance of the feldspars it should be further designated as an arkose sandstone. And considering the extent to which it has been modified by the development of interstitial siliceous products and the effect that this has had in perfecting the bond between the grains, the rock may be classified as an indurated arkose sandstone.

Special Structure. — A study of the cause of reeding, or the tendency to split into slabs, led to the preparation of thin sections of this structure (Fig. 1). It is apparent from them that the reed is strictly a rock structure and that the perfection of the capacity to split along these planes depends wholly upon the abundance and arrangement and size of the elongate and semi-fibrous grains and the presence of a more than usual amount of original fine or flaky material. Almost universally the reed streaks are darker in color and finer in grain than the average of the rest of the rock.

In part therefore it is an original character due to the assorting action of water during deposition, finer streaks alternating with coarser ones in accord with ordinary sedimentation processes. But, in addition to that, the subsequent changes that have affected the whole rock have occasionally accentuated the structure by a tendency of the whole rock to develop elongate or fibrous aggregates. It is probable therefore that the parting capacity is in places considerably increased by the very process that has produced just the reverse results in the more heterogeneous portions of the beds.

Under a sufficient stress the rock will part most easily along the planes where this foliate or fibrous character is most persistent. Even in these cases, however, it may not indicate that the rock is essentially weak. It simply locates the most vulnerable point in the stone. In many quarries these streaks are so abundant that only thin slabs can be obtained — the disturbances of ordinary quarrying being sufficient to cause parting. The deeper portions of quarries are however much less subject to such behavior. In all cases the greater slab development of the exposed portion of

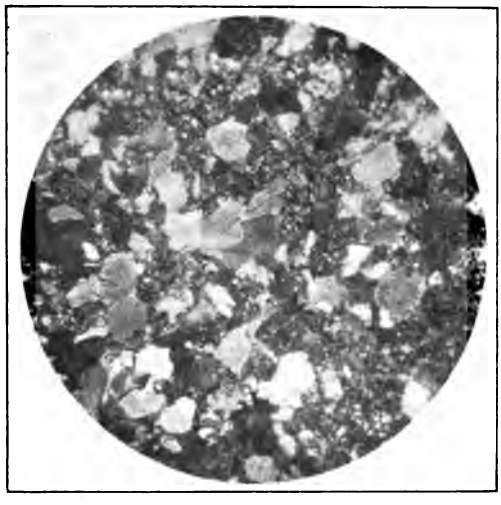


FIG. 3. Photomicrograph of medium grain bluestone of first grade, — × 25 diameters. Shows angular and interlocking grains taken to be evidences of secondary growth. There is a complete lack of reed structure. The clear grains are quartz, the rest of the field is occupied chiefly by the secondary derivatives of original feldspars.

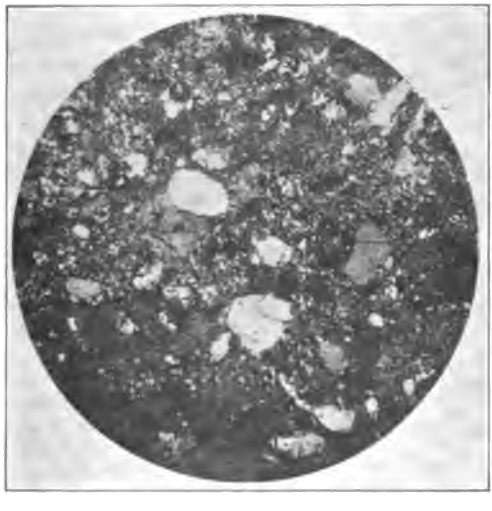


FIG. 4. Photomicrograph of bluestone, — × 25 diameters. The clearer grains are quartz and indicate the approximate size of the other original constituents. In this case the change of feldspars is so complete that their products form a complex aggregate of closely interlocked grains and fibers of extremely fine texture.

the ledge is an ordinary weathering effect, by which the same results are obtained slowly and naturally and more perfectly than can be secured artifically on the fresh material of the same beds. The expansions and contractions of changes of temperature, together with the rupturing effects of freezing water caught in the pores, serve finally to weaken every part of the rock. In this process the prominent reed lines give way so much in advance of the rest of the rock that they develop into true rifts and separate slabs appear. It must be appreciated that these ledges have been exposed an immensely long time compared with the probable requirements of any engineering structure, and that this weathering tendency does not mean a speedy disintegration of the freshly quarried blocks. Still it is advisable to avoid as many sources of weakness as possible and one of the ways is to select ledges where the stone does not have a reeding tendency, or in which the reed lines are interlocked, or wavy, or interrupted. These requirements are most fully met in the coarser beds and especially those exhibiting some cross-bedding. Two of the quarries named in Part I. meet these demands to a marked degree.

Strength. — The better qualities of bluestone have great strength. Even the reed lines are in many instances stronger and more durable than the regular quality of some other sandstones that are usually considered suitable building material. The secret of this exceptional strength lies in the modifications of texture that have resulted from the alteration and reconstruction of the mineral constituents. The breaking up of the orthoclase feldspar, and the accompanying changes in the ferro-magnesian minerals, have furnished considerable secondary quartz, which has in part attached to the original quartz grains making them more angular and developing an interlocking tendency (Fig. 3). At the same time the fibrous sericitic and chloritic aggregates have developed to such extent as to fill most of the remaining pores, and in many cases the fibrous extensions have actually grown partly around the adjacent quartz grains (Fig. 2). The effect has been to develop a siliceous binding of unusual toughness. This combination of changes has made a rock that is now remarkably well bound or interlocked for a sedimentary type.

Durability. — First class stone of the grades indicated above would have as great durability as any stone in the market, except perhaps a true quartzite. With the exception of the almost neglect-

able quantities of pyrite, occasionally found, there is no constituent prominently susceptible to decay. The rock as a whole mineralogically is stable and its texture indicates unusual resistance to ordinary disintegrating agencies.

Conclusion. — If the ledges, as pointed out, upon further exploration, prove to be able to furnish sufficient quantities of stone of the grades recommended, it would be wholly unnecessary to make extensive examination of more distant localities. For no better stone need be desired for all of the proposed uses than the bluestone of the immediate vicinity of the proposed Ashokan Dam.

ROAD RESISTANCES.

By C. E. MORRISON.

Since in transportation of any kind, whether by means of cars as on railways, or wagons as on ordinary roads, the object is to haul a load as cheaply as possible, that is, aside from the questions of initial cost and maintenance and repair, to haul the greatest load with the least expenditure of energy, it will be quite proper in the subject of roads and pavements, to consider first, those factors tending to affect the tractive force per unit load, and thus the cost of transportation.

By tractive force is meant the force exerted by a team in drawing a load over a road, and it is usually expressed in pounds per ton. The resistances to be overcome, called tractive resistances, are due to the load itself and the wagon carrying it. The general term tractive resistance as applied to wagon haulage, may be separated into three parts: (1) Axle friction, (2) Rolling resistances, and (3) Grade resistances, and from this consideration the most advantageous condition of road transportation may be determined.

Axle Friction. Not very much is definitely known concerning this factor in wagon haulage, not only because of the few experiments made for its determination, but because of its relatively small importance as compared with the other resistances. Those experiments that have been made, however, seem to check fairly well similar values obtained with railway car journals and in machines.

Axle friction depends upon the nature of the rubbing surfaces, *i. e.*, the material of which the hub and axle are made, upon the degree of lubrication, and the nature of the lubricant. With poor lubrication, this factor will be from two to six times the value obtained with good lubrication. In car journals, at least, the axle friction is dependent to some extent upon both velocity and temperature. That it varies inversely as some function of the pressure is known but the exact relation has never been determined. According to Baker, for light carriages when loaded, the coefficient of friction is about 0.020 of the weight on the axle, for heavier carriages 0.015, and for common American wagons 0.012; or for

the above carriages, from 3 to 4½ pounds of tractive force per ton of weight on the axle may be charged to axle friction. As the diameter of the axle is decreased, and that of the wheel increased, the friction resistance may be diminished, but it should be observed that an undue decrease in the cross-section of the axle increases the axle pressure per unit area, causing excessive heat and wear.

In train resistances, the coefficient of resistance in pounds per ton due to axle friction, has been found to be equal to, the product of the coefficient of friction times the diameter of the axle, divided by the diameter of the wheel, multiplied by the number of pounds in a ton — *i. e.*, 2,000 or 2,240, or

$$\text{Coefficient of resistance} = \frac{\text{coefficient of friction} \times \text{diameter of axle}}{\text{diameter of wheel}} \times 2{,}000.$$

ROLLING RESISTANCES. This is dependent upon several factors, but it is due to the fact that no matter how perfect the roadway surface the wheel of a vehicle will always sink to some extent into it in consequence of the elasticity of the material, and the wheel is thus always in the act of rolling up a small incline. This of course is less in the case of the wheels of a locomotive on steel rails than in that of an ordinary cart hauled over an average road, but it exists in the former as well as in the latter.

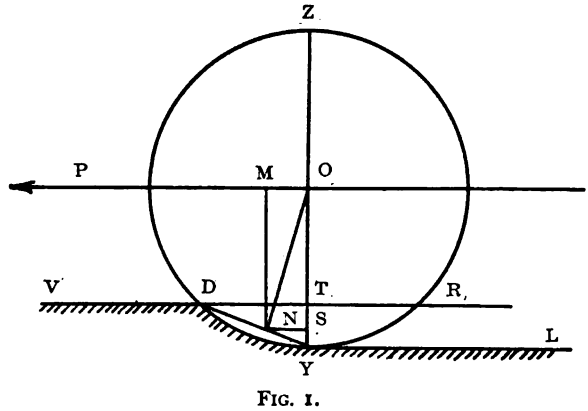


FIG. 1.

The tractive force required to overcome this resistance might be found by the following demonstration. In Fig. 1, let *VDR* be the

original and natural surface of the road, O the center of the wheel ZYD, TY the depth to which the wheel has sunk into the road metal, and P the direction of the motive force.

The submerged portion of the wheel is DY, of which it may be assumed that the arc corresponds with the chord. The resistance to penetration then is a maximum at Y, zero at D, diminishing with the depth of depression, and may be represented by an isosceles triangle with the center of gravity at N, located one third the distance YN from the base Y. This is also the center of resistance and pressure of the load measured in direction and intensity by OS. The tractive force is measured by NS or MO. Since the depth TY is small, OS may be assumed equal to the radius of the wheel OY, and NS as one third of the one half chord DT. It follows then that if W = the load, T.F. = the tractive force, and R = radius of wheel,

$$W : \text{T.F.} :: OY : \frac{DT}{3} :: R : \frac{DT}{3}$$

and

$$\text{the resistance to traction} = \frac{W \times \frac{DT}{3}}{R}.$$

but

$$DT = \sqrt{ZT \times TY}$$

and therefore the tractive force,

$$MO = \tfrac{1}{3} \times \frac{W\sqrt{ZT \times TY}}{R}.$$

On the other hand the power required to draw a wheel over an obstacle such as a stone may be determined as follows. Let O be the center of the wheel, YZ the road, S the obstacle, and P the direction of the motive force OM. This is the force necessary to draw the wheel to the point S. The moment of the tractive force is equal to the force OM times its lever arm MS. But the weight of the load is acting in the direction OY and is represented in intensity by OX: its lever arm is therefore equal to XS. For a condition of equilibrium $XS \times OX = OM \times MS$. If OM be represented by P and OX by W, then the tractive force

$$P = W\frac{XS}{MS}.$$

The various components of rolling resistance affecting its value are: (*a*) The diameter ot the wheel; (*b*) The width of the tire; (*c*) The speed; (*d*) The presence or absence of springs, and (*e*) The road surface.

The Diameter of the Wheel. Experimentally it has been determined that the diameter of a wheel affects the rolling resistance in

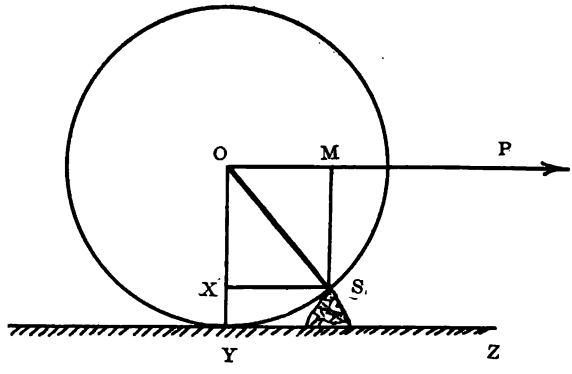


FIG. 2.

some inverse ratio. This may be accounted for by the fact that the smaller wheel makes a deeper depression in a road than the larger and therefore requires greater effort to haul it out. The previous figure gives the following solution:

$$P \times MS = W \times XS \quad \text{or} \quad P = W\frac{XS}{MS}$$

$$XS : MS :: \tan XOS : 1 \quad \text{(dividing by } MS\text{)}$$

$$\tan XOS = \tan 2(SYZ)$$

$$\therefore P = W \tan 2(SYZ).$$

But as the SYZ increases inversely as the diameter of the wheel, the value of P will vary in like manner, and large wheels are hence better than small ones.

M. Morin experimenting upon the effect of diameter found that on a given road and with a given load, a 6 ft. 8 in. wheel had but little effect upon the road surface; that a 4 ft. 9 in. wheel cut deep ruts, and that a 10.5 in. wheel cut still deeper ruts.

At Bedford, England, in 1874, it was shown what effect the

diameter of a wheel exerted upon the tractive force, when it was demonstrated that a pull of one pound could move a load of 35.1 pounds resting on a wheel 3 ft. 5 in. in diameter, whereas the same force could move as much as 58.7 pounds on a wheel 5.0 ft. in diameter.

Naturally the smaller wheels present the smaller bearing surface to the road, thus increasing the pressure per unit area for equally weighted vehicles. This increase frequently causes the depressions in, and undue wear of, the road's surface.

The Width of Tire. Perhaps no one of the above-mentioned components of rolling resistance has been discussed so frequently and so much as that of width of tire, in its effect upon both haulage and road preservation. Generally the subject is approached from the standpoint of road preservation due to wide tires, rather than that of decrease in tractive force, but both are worthy of careful consideration.

Narrow-tired wheels, like those of small diameter, are frequently required to carry a greater load per unit of bearing surface than is desirable, with the consequent effect that they not only increase the effort upon the horse, but sinking deeper into the road, tend to destroy it more quickly.

Though it is generally considered that the adoption and use of broad tires will improve the road surface, the fact was mentioned by the Massachusetts State Highway Commission in its Report of 1893, that results might not warrant such a change. Assuming that in the state of Massachusetts there were 50,000 vehicles requiring a change in the width of tire, at an average cost of $20 a vehicle, this would mean an expenditure of $1,000,000 for the alteration. It was pointed out that with this amount, 200 miles of very excellent road could be built "of the kind that would not be affected by any width of tire." This is a point well taken, but it should be remembered that varying conditions require different consideration.

In New Jersey it is maintained by the commission that when a road is improved, immediately heavier loads are hauled on the same tires, causing ruts which hold water, with the consequent result of the disintegration of the foundation. Such ill effects can only be overcome by wider tires, thus decreasing the unit load, and the destructive effects of the wheels. It has also been suggested that to supplement the good effect of road-rolling machines,

4 inch tires be required on heavy trucks and that their owner's license be denied until such requirements be complied with.

To this effect, an act of the New Jersey Legislature dated March 24, 1896, permits township committees to pass an ordinance allowing a rebate of taxes to owners or possessors of wagons and carts with tires not less than four inches in width; and in Maryland, on toll roads, only one half the rate is collected from drivers of broad-tired wagons.

Experiments show that with a uniform load and wheels of equal diameter, a tire 2.4 inches causes double the wear to the road that is produced by a tire 4.6 inches wide; but it may be questioned if tires broader than this have any advantage.

Tests made at the Missouri Experiment Station covering a period of two years, on macadam, gravel and dirt roads, in all conditions, and on meadow, pasture and plowed land, both wet and dry, show the following results. The tests were made with a net load of 2,000 pounds in every case, a self-recording dynamometer being used to measure the tractive force.

"1. *On Macadam Street.* As an average of two trials made, a load of 2,518 pounds could have been hauled on the broad tires (6 inches) with the same draft that a load of 2,000 pounds required on narrow tires (1½ inches).

"2. *Gravel Road.* In all conditions of the gravel road, except wet and sloppy on top, the draft of the broad tired wagon was very much less than that of the narrow-tired wagon. Averaging the six trials, a load of 2,482 pounds could be hauled on the broad tires with the same draft required for a load of 2,000 pounds on the narrow tires.

"3. *Dirt Road.* (*a*) When dry, hard and free from ruts and dust, 2,530 pounds could have been hauled on the broad tires with the same draft required for 2,000 pounds on the narrow tires. (*b*) When the surface was covered with two or three inches of very dry loose dust, the results were unfavorable to the broad tire. The dust on the road in each of these trials was unusually deep. (*c*) On clay road, muddy and sticky on the surface and firm underneath, the results were uniformly unfavorable to the broad tires. (*d*) On clay road, with mud deep and drying on top, or dry on top and spongy underneath, a large number of tests showed uniformly favorable to the broad tire. The difference amounted to from 52 to 61 per cent., or about 3,200 pounds could have been hauled on the broad

tires with the same draft required to draw 2,000 pounds on the narrow tires. In this condition of road the broad tires show to their greatest advantage. As the road dries and becomes firmer, the difference of the draft of the broad and narrow tires gradually diminishes until it reaches about 25 to 30 per cent. on dry, level, smooth dirt, gravel or macadam road in favor of the broad tire. On the other hand, as the mud becomes softer and deeper, the difference between the draft of the two types of wagons rapidly diminishes until the condition is reached when the mud adheres to both sets of wheels; here the advantage of the broad tires ceases entirely, and the narrow tires pull materially lighter. (*e*) Clay road, surface dry, with deep ruts cut by narrow tires in the ordinary use of the road. In every trial the first run of the broad tire over the narrow-tire ruts showed a materially increased draft when compared with that of the narrow tire run in its own rut. The second run of the broad tires in the same track, where the rut is not deep, completely eliminated this disadvantage, and showed a lighter draft for the broad tire than the narrow tire showed in the first run. Where the ruts were eight inches deep, with rigid walls, three runs of the broad tire in its own track over the ruts were required to eliminate the disadvantage. Three runs of the broad tire over this track have in all cases been sufficient, however, to so improve the road surface that both the broad and narrow-tired wagons passed over this road with less draft than the narrow tires did in the original ruts. In addition to the saving of draft, the road was made very much more comfortable and pleasant for the users of light vehicles and pleasure carriages by the few runs of the six-inch tire. Summing up all the tests on dirt roads, it appears that there are but three conditions on which the broad tires draw heavier than the narrow tires, namely, (1) When the road is sloppy, muddy or sticky on the surface, and firm or hard underneath; (2) When the surface is covered with a very deep, loose dust, and hard underneath; (3) When the mud is very deep and so sticky that it adheres to the wheels on both kinds of wagons. It appears that the dust must be extraordinarily deep to show a higher draft from the broad tires than from the narrow tires. The three conditions just named, therefore, are somewhat unusual and of comparatively short duration. Through a majority of days in the year, and at times when the dirt roads are most used, and when their use is most imperative, the broad-tired wagons pull materially lighter than the narrow-

tired wagons. (4) A large number of tests on meadow, pasture and stubble-land, corn ground and plowed ground in every condition, from dry, hard and firm to wet and soft, show without a single exception, a large difference in draft in favor of the broad tires. This difference ranged from 17 to 120 per cent. (5) It appears that six inches is the best width of tire for a combination farm and road wagon, and that both axles should be the same length so that the front and hind wheels will run in the same track."

It is therefore evident that wide tires not only tend to diminish the draft under most conditions, but that they also aid in the preservation of the road surface.

Speed increases the rolling resistances as the shocks, due to irregularities in the surface, occur with greater frequency. On a uniformly smooth surface, MM. Morin and Dupuit found that the resistance to traction is independent of the speed, but on ordinary roads or pavements it has been shown to vary as some power of the velocity. Naturally it is dependent upon the surface and also upon the presence or absence of springs.

Springs decrease the shock to which the vehicle is subject in rolling over rough surfaces and hence diminish the tractive force as well as wear and tear to wagon and road. For smooth roads, this will vary less than for rough. Like speed, it is but a small factor in rolling resistance.

Road Surface. The effect of the road surface upon rolling resistance and tractive force is quite evident in the two extreme cases of a railway and an ordinary dirt road. It requires no demonstration to be convinced that for the same expenditure of energy, other things being equal, a very much heavier load may be hauled on the former than on the latter. In other words, the smoother and harder the road for a level surface, and disregarding the question of foothold, the greater the load that may be hauled for a unit of power. Smoothness and hardness not only reduces tractive force, but produce comfort, and diminish shock, wear and tear on road, vehicle, and animal.

The following figure represents graphically the conditions resulting from various pavements.

As showing the effect of surface upon tractive force the following table from Sir John McNeill is given:

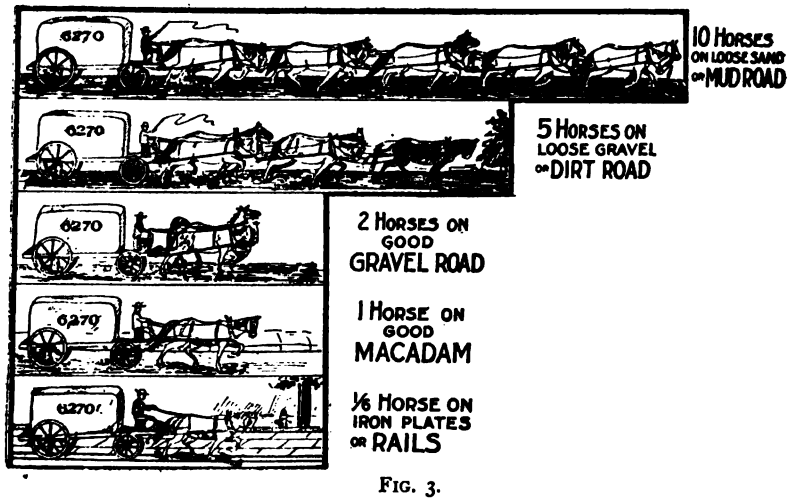

10 HORSES ON LOOSE SAND OR MUD ROAD
5 HORSES ON LOOSE GRAVEL OR DIRT ROAD
2 HORSES ON GOOD GRAVEL ROAD
1 HORSE ON GOOD MACADAM
⅙ HORSE ON IRON PLATES OR RAILS

FIG. 3.

Kind of Road.	Tractive Force. Pounds per Ton.
Well made block pavement	33
Telford or macadam	46
Old flint road	65
Loose gravel	147

The results tabulated below are from the tests made by M. Morin:

Kind of Road.	Tractive Force.			
	Carts, 3 Miles per Hour. Pounds per Ton.	Wagons 2 Tons, 3 Miles per Hour. Pounds per Ton.	Carriages with Springs, 3 Miles per Hour. Pounds per Ton.	Carriages with Springs, 6 Miles per Hour. Pounds per Ton.
New road with five inches of gravel	166	222	250	
Earth road with one half inch of gravel	125	191	200	
Broken stone in best condition	27	37	42	48
Broken stone very much rutted and very muddy	105	143	166	200
Pavement, clean	11	30	35	53
Pavement, muddy	14	40	45	66

The following table from Rudolph Herring shows the effect of road surface on tractive force. The velocity in each case is three miles per hour.

Kind of Road.	Tractive Force. Pounds.	Authority.
Loose sand	448	Bevan.
Loose gravel (deep)	320	Do.
Loose gravel (4 inches)	222	Morin.

Common gravel road		147	McNeill.
Good gravel		88	Rumford.
Hard rolled gravel		75	Minard.
Ordinary dirt road		224	Bevan.
Hard clay		112	Do.
Hard, dry, dirt road		89	Morin.
Macadam, little used	140 to	97	Do.
Bad macadam		160	Gordon.
Poor macadam		112	Navier.
Common macadam		64	Peidounet.
Good macadam, wet	75 to	42	Morin.
Best French macadam		45	Navier.
Very hard and smooth macadam		46	McNeill.
Best macadam		50	Rumford.
Do.	49 to	37½	Gordon.
Do.	52 to	32	Morin.
Cobblestone, ordinary		140	Kossack.
Cobblestone, good		75	Do.
Belgian block		56	McNeill.
Belgian block in Paris	54 to	34	Navier.
Belgian block		37½	Rumford.
Belgian block, good		34½	McNeill.
Belgian block	50 to	26	Morin.
Stone block, ordinary		90	Minard.
Stone block, good		45	Rumford.
Stone block, London		36	Gordon.
Asphalt		17	Do.
Granite tramway		13½	Do.
Do.		12½	Do.
Iron railway		11½	Do.
Do.		8	Lecount.

The results of traction tests given below were made by the Office of Road Inquiry of the U. S. government.

Kind of Road.	Tractive Force. Pounds.
Loose sand (experimental)	320
Best gravel (park road)	51
Best clay	98
Best macadam	38
Poor block pavement	42
Cobblestone	54
Poor asphalt	26

In the line of relative tractive resistance on brick and asphalt pavements few tests have yet been made, so that the following results obtained from tests made by the Department of Civil Engineering of Iowa State University are of some value.

TRACTION TESTS ON BRICK AND ASPHALT PAVEMENTS.

Character of Pavements.	Traction Resistance. Pounds per Ton.
Brick Pavement.	
One course, sand filled, much worn and broken	58
On concrete, sand filled, much worn and rough	43.3
Same as above	37.6
On concrete, sand filled, dirty, well worn	33.3
On concrete, nearly new, cement filled	31.7
On sand, sand filled, condition fair	29.1
New, sand filled	37.2
Old, tar filled, smooth, clean	28.2
Old, cement filled, clean, smooth	25.4
Clean	16
Dirty	19
Asphalt Pavement.	
New, quite dirty	26.7
New, clean, smooth	24.0
New, clean, smooth	23.3
Old, clean, much used	29.1
New, clean, little used	24 2
Old, dirty, little used	39.9
Old, clean, very little used, pitted	67.8
Same as above, but colder	36.5
Same as above after period of cold weather, dirty	34.4
Clean, new, very little used	55.5
Same as above but colder	31.1
Same as above after period of cold weather, fairly clean	26.5
Block asphalt, one year old, much used, worn, clean	28
Clean	34

The above were all made with ordinary draft wagons.

Traction Tests. The determination of values such as given above constitutes what is known as a traction test. Various instruments have been used to measure the amount of effort exerted by a horse in hauling a load over a road, with the object of learning which pavement permits of the largest load, and hence gives the cheapest cost of transportation.

In road construction, as in every other branch of engineering, it should be remembered that the question of cost is important — if not the most important — and that good roads are represented by those over which loads may be transported for the least expenditure. To compare the cost of haulage it therefore becomes necessary to determine the effort exerted. The pioneers and authorities along this line were M. Morin who experimented for the French government, and Sir John McNeill in England. The lat-

ter's results were published in 1848, while the former's appeared in 1843.

The more recent results of the experiments of the Department of Agriculture, made in 1895, are of distinct value. They were made on the roads of the United States Road Exhibit at the Cotton States and International Exposition in Atlanta, Ga., in 1895. The wagon and load was similar to those in daily use in the southern states to demonstrate to the public the advantage of improved roads. Three kinds of roads were used, a modern macadam, a sand road, and an ordinary dirt road, the latter two being types of the more common roads in the south.

The macadam road was 300 feet long, made up of six 50-feet sections, varying in rise by two per cent. increments, from a level to a grade of 10 per cent. The sand and the dirt roads were each 200 feet long, divided into 50-feet sections, varying in a similar manner from a level to a grade of 6 per cent. The tests therefore show the results not only of road surface, but grade as well.

The macadam road, 12 feet wide, had a natural clay sub-grade, which was well rolled. On top of this 4 inches of 2-inch Blue Trenton Limestone was placed, followed by a coarse of 2-inch screenings, each being subjected to the action of a road roller.

The sand road was 12 feet wide also, consisting of 6 inches of river sand placed on a bed of natural clay, neither the bed nor the surface being rolled.

The dirt road was made of natural earth. To approximate general conditions, it was thoroughly wetted and had a heavily laden narrow tired wagon passed over it to form ruts and depressions.

To measure the force applied to haul the wagon over these roads with varying grades, a simple modification of the spring balance was used with a large index finger moving over a graduated arc to show the pull in pounds and having one end attached to the wagon and the other to the team. The record was simply by the eye, and showed that even for the smoothest possible macadam the force of traction was not constant, but changed continually within a limit of 50 pounds. With dirt roads the force varied from zero to 700 pounds in a gross load of 3,000 pounds, and showed that it was applied in a manner to represent a succession of rapid and violent jerks. On heavy, smooth grades the force was more nearly constant, as might be expected; while to

start the load under any conditions it required about four times the force necessary to haul it.

From these preliminary observations the following conclusions were drawn: On the smoothest road a team is subject to a jerking motion which greatly fatigues. This is greatest on a dirt road because the smoother the surface the less the effect of such action. It requires about four times the effort on a level dirt road, as upon a level macadam. Hard roads reduce the tractive force, and sand roads increase the pull enormously.

Supplementing these tests a tractograph was used to determine and record the force applied. The figures below show it in position on the shaft of the wagon. It consists of an arm, placed longi-

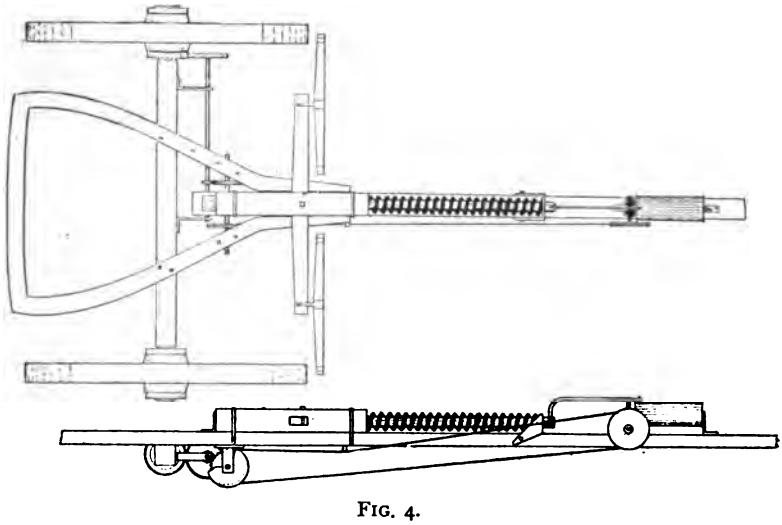


FIG. 4.

tudinally along the pole, holding a pencil point resting on a stationary cylinder covered with a sheet of paper. By means of the connections shown, the cylinder was made to revolve so that it completed one revolution in a distance of 1,316 feet. The spring was standardized against known weights varying by 100 pounds from zero to 1,200, and hence the compression of the spring per unit became known. From these facts it was possible to have a sheet of paper divided into spaces corresponding to pulls of ten pounds, and into other spaces corresponding to distances of 100 feet, which made known the exact amount of pull and the place at which it was applied. The compression of the spring due to a

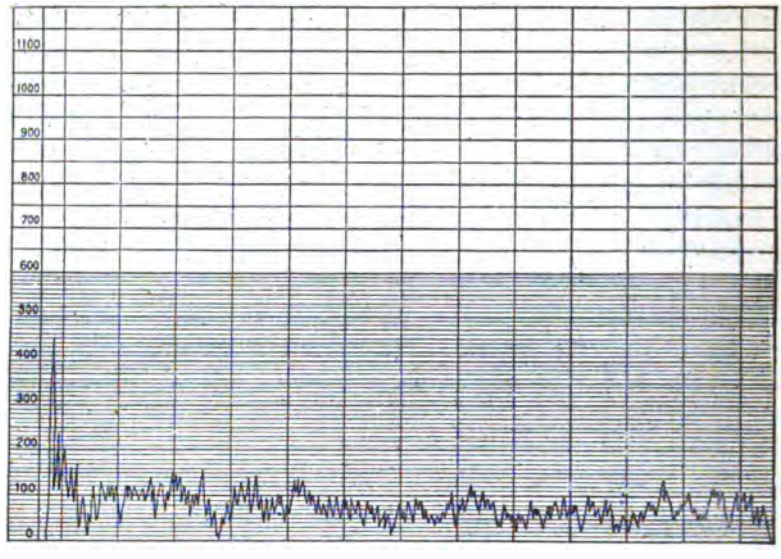

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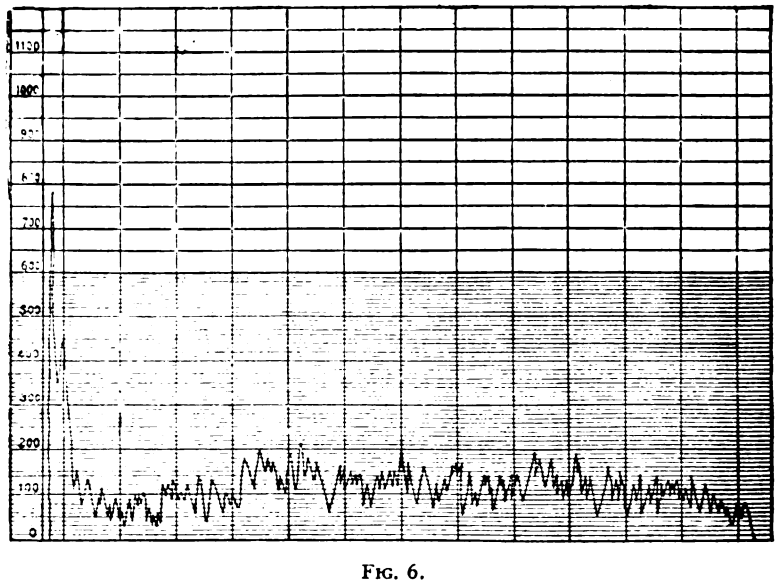
FIG. 5.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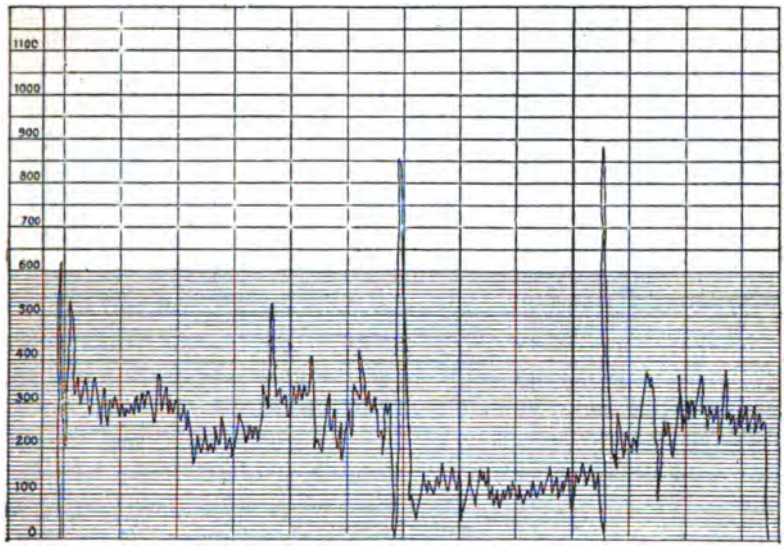
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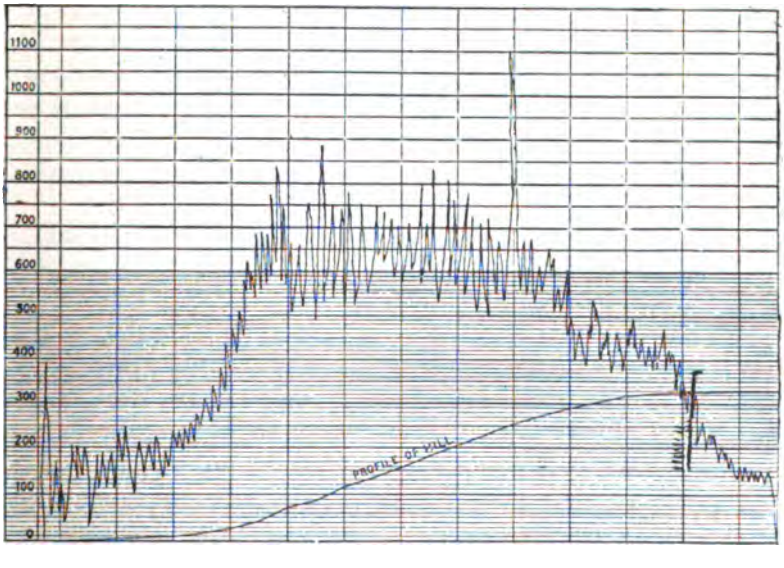
FIG. 7.


1100
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900
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700
600
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300
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PROFILE OF HILL

FIG. 8.

pull on the double-trees caused the pencil point to move forward on the cylinder, and thus make the record. The diagrams following show the records obtained: Fig. 5, on an asphalt street; Fig. 6, on a smooth macadam road in good condition; Fig. 7, on a smooth dirt road, and Fig. 8, on a macadam road ascending a hill of 10 per cent. grade.

The asphalt was not in good condition, being rough and uneven, as is quite conclusively shown by the irregularity of the record.

The macadam pavement was in excellent condition.

The dirt road was also in good condition; dry, firm and smooth. It crossed the macadam pavement in two places, and this is shown by the two high points in the curve, being due to the fact that the team stopped on either side of the pavement.

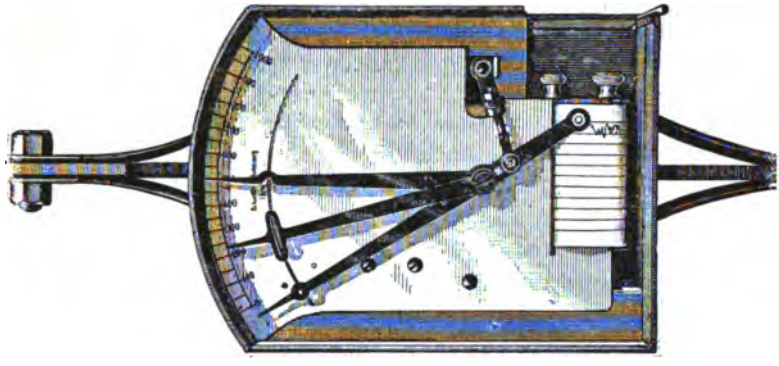
Fig. 8 shows the effect of a steep grade upon the tractive force, and under the existing conditions the extremely irregular curve is due to the fact that the team required continual urging to ascend the hill at all. Compared to the diagram of the level stretch, it indicates that about seven times the force was required on the former as upon the latter.

Under all conditions, the high points of the curves at the start indicate that from four to eight times the effort to haul the load was required to set it in motion.

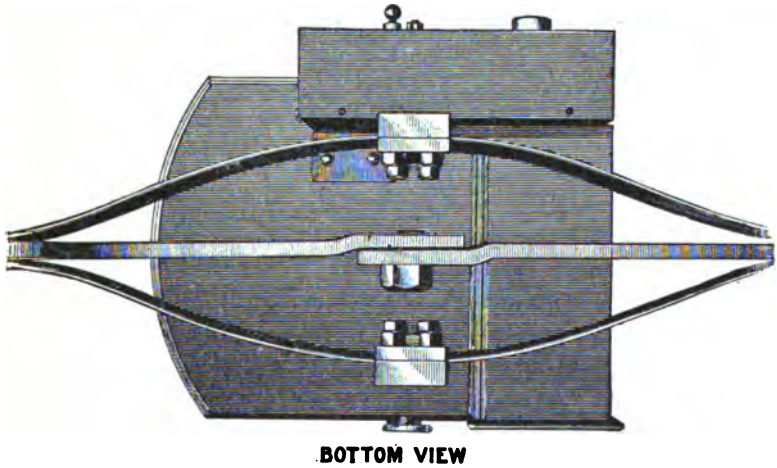
If the average amount of tractive force for each road is desired, it may be computed by taking the area between the zero line and the curve and dividing by the distance. From such calculations it was found that the smooth asphalt street required 26 pounds per ton, the smooth macadam road 38 pounds per ton, the smooth dirt road 96 pounds per ton, and on the 10 per cent. grade 136 pounds per ton.

A similar instrument for measuring the tractive force and recording it automatically is the Baldwin dynograph, a cut of which is shown below.

This instrument consists essentially of two flat springs with ends connected, and centers separated from each other by the small distance shown. One end of the springs is attached to the wagon and the other end to the team. Any force exerted by the team tends to draw the centers of the springs together and this contraction is recorded, first by the small pencil point moving over the revolving cylinder which carries a graduated sheet of paper, and second by two index arms which show against the scale the maximum number of pounds pull, and also an approximate average.



TOP VIEW



BOTTOM VIEW

FIG. 9.

GRADE RESISTANCES. Aside from the question of drainage, the ideal road should be as nearly level as possible, since under these conditions it favors the load neither going or coming, and presents less resistance to traction.

To calculate what these resistances are, let us assume in Fig. 10 that G is the center of gravity of the load; then GA is the direction and amount of the weight, while BG represents the force pressing against the inclined surface DE. The line EF represents the rise per unit of length DF, which is the horizontal. From the two similar triangles we have $GA : AB :: DE : EF$; but GA is the weight of the load, DE is the length of road and EF is the rise.

Representing these by W, l and r, we have,

$$W : AB :: l : r \text{ or } AB = W\frac{r}{l}$$

but AB is the resistance due to grade.

Or we may write the formula

$$R = F + aW.$$

Where R = force required to draw the load on an incline,
F = force required to draw the load on the level,
a = the grade expressed as a fraction,
and W = the weight of the load in pounds.

Regarding the effect of grade, the Massachusetts State Highway Commission in its Report for 1901, says: "By far the most serious defect in the old town highways is the heavy grades. These are

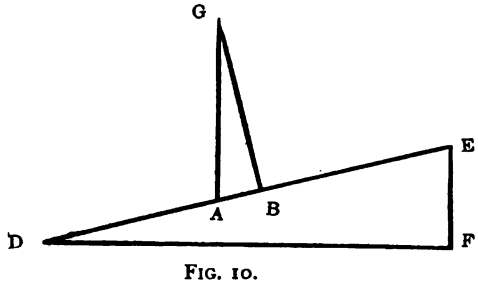


FIG. 10.

not only a tax on the user, but they are a constant and burdensome cost to the municipalities having to care for them. Taken as a whole no one improvement makes a better return for the money invested than the cutting down of hills. Whatever is done in this direction is a permanent benefit, as by carefully grading a road throughout its entire length between centers of population or business, the possible load is increased, notwithstanding no particular effort be made to improve the surface."

The table below shows the effect of grade on the best macadam roads in the variation of the tractive force.

Rate of Inclination.	Tractive Force. Pounds per Ton.
Level	38
1 in 500 = 0.2 of 1 per cent	42
1 in 100 = 1.0 per cent.	58
1 in 80 = 1.25 per cent.	63
1 in 60 = 1.66 per cent.	71
1 in 50 = 2.0 per cent.	78

1 in 40 =	2.5 per cent.	88
1 in 30 =	3.33 per cent.	104
1 in 25 =	4.0 per cent.	118
1 in 20 =	5.0 per cent.	138
1 in 15 =	6.66 per cent.	171
1 in 10 =	10.0 per cent.	238

This is graphically represented in the following cut.

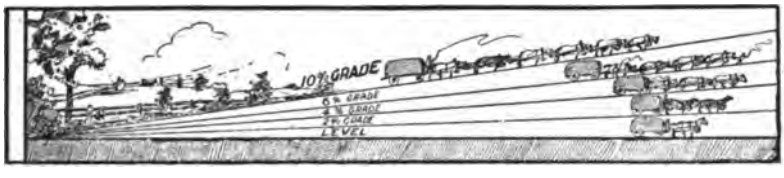


FIG. 11.

In any case the maximum grade is that which determines the load that may be hauled over any section of a road, and hence it should always be the object to reduce grade to a minimum. Furthermore, a horse's effective power for hauling, rapidly diminishes going up inclines, since it is compelled to overcome the increased resistance of gravity due to its own weight, as well as that due to the load itself. It should be taken into consideration, however, that for a short distance a horse is able to exert several times its normal tractive pull, and it should be remembered also, that the smoother and harder the surface the poorer may be the foothold, and consequently the less the power that a horse may exert.

This is quite conclusively demonstrated by the following table taken from Baker, showing the effect of grade and road covering upon the load a horse may draw, the load being in terms of weight of horse:

Grade Per Cent.	Iron Rails.	Sheet Asphalt.	Broken Stone.	Stone Block.	Earth Road.		
					Best.	Spongy.	Muddy.
0	20.00	10.00	6.00	5.00	3.00	2.00	1 00
1	6.00	4.50	3.33	2.00	2.09	1.50	0.91
2	3.20	2.67	2.16	2.00	1.51	1.14	0.67
3	2.00	1.75	1.49	1.40	1.11	0.87	0.54
4	1.33	1.20	1.05	1.00	0.82	0.66	0.43
5	0.91	0.83	0.75	0.71	0.60	0.50	0.33
6	0.62	0.59	0.52	0.50	0.43	0.36	0.25
7	0.40	0 38	0.34	0.33	0.29	0.25	0.18
8	0.24	0.22	0.21	0.20	0.18	0.15	0.11
9	0.15	0.10	0.09	0.09	0.08	0.07	0.05
10	0.00	0.00	0.00	0.00	0.00	0.00	0.00

Acknowledgment must be made for information taken from the following sources, for which no credit is given in the text: The various State Highway Commissions, Massachusetts, Connecticut, Rhode Island, New York, New Jersey, and Maryland; the various reports relative to the subject issued by the U. S. Department of Agriculture; and to the authors of works on highways, such as Byrne, Baker, Judson, Aitken, Tillson, etc.

FUSION TABLE OF MINERALS IN THE OXY-GAS BLOWPIPE FLAME.

By LEA McI. LUQUER.

During the past months some experiments have been conducted on the effects of high temperature on the crystallization of certain chemical compounds. These laboratory experiments were made on small amounts of the compounds and the high temperatures obtained with the oxy-gas blowpipe. The results have suggested to the writer the following series of tests.

In nearly all practical schemes for the determination of minerals by fusion and solubility tests, there are quite a number of the common minerals "left over" in the fusion classification and placed in a large division of the scheme, under the head "infusible in the blowpipe flame."

These minerals when tested with the oxy-gas blowpipe flame yielded interesting results, which not only serve as records of the effects of high temperature, but which could be used to form an auxiliary scheme for the practical determination of the so-called "infusible minerals."

For these fusion tests an ordinary gas blowpipe nozzle was used, which for convenience was mounted on an universal jointed stand, about four inches high with a heavy base. Connections were made with the gas pipe and oxygen cylinder by means of rubber tubes. A rather low gas flame was used, which, after the turning on of the oxygen blast, lengthened out, with a slight hissing sound, into a blue flame about one to one and a half inches long. The hottest part of the flame was found to be at the point of an inner blue cone very close to the end of the nozzle. In these tests platinum (melting point 1753° C.) was melted easily, corundum (melting point 1865° C.) and spinel (melting point 1900° C.) fairly easily, therefore the temperature must have been approximately between 1900° and 2000° C.

It was intended at first to employ crushed fragments of the minerals, of uniform size, but this was not practicable on account of the difficulty of preventing the fragments from being blown away by the blast and also because of the rapid combustion or

FUSION TABLE.

(A sign + indicates the tests obtained.)

Mineral.	Fusion.			Not Fused.	Bead or Glass Produced.				Remarks.
	Easy.	Fairly Easy.	With Difficulty.		Clear and Colorless.	White	Colored	Black.	
Graphite, Ticonderoga.				+					Easily consumed
Molybdenite, Ontario.		+						+	Volatilizes with white fumes and yellow-green flame. Black sinter formed on edge.
Quartz (colorless crystal).	+				+				
" (white massive).	+				+				Bead contains a few bubbles.
" (rose).	+				+				Loses color.
" (amethyst).	+				+				Loses color.
" (flint).	+					+			Loses color, turns white. Bead filled with bubbles.
Corundum (light sapphire), Ceylon.		+					grayish		Bead has rough, bubbly surface.
Corundum (light ruby), Ceylon.		+					light pink		Loses color slightly. Bead has rough, bubbly surface.
Corundum, Craigmont, Can.		+						+	Loses color. Bead has rough, bubbly surface.
Ilmenite (Washingtonite), Litchfield.	v. easy							+	Crystalline bead. Scintillates.
Cassiterite, Zinnwald.		+						+	Coated with white oxide, which is turned green with nitrate of cobalt.
Rutile, Alexander Co., N.C.	+							+	
Rutile (Nigrine), Magnet Cove.	+							+	
Brookite, Diamantine.	+							+	
Pyrolusite, Ga.	v. easy							+	Bead contains bubbles.
Opal (semi-opal), Silesia.	+					+			Decrepitates, whitens.
Opal (Hyalite).	+					+			Bead contains bubbles.
Diaspore, Chester, Mass.	+							+	Whitens.
Manganite, Thuringia.	v. easy							+	Scintillates.
Gœthite (needle crystal), Cornwall.	v. easy							+	
Limonite.	+							+	May decrepitate. Scintillates.
Bauxite (light colored), Baux.	+						brown		Sputters.
" (whitish), Polk Co., Ga.	+						gray-blk		
" (reddish), Ark.	+							+	
Brucite, Texas, Pa.				+					Splits, whitens, glows intensely.
Gibbsite, Richmond, Mass.	+					+			
Gibbsite.	+						grayish		Sputters.
Calcite (transparent cleavage).			on edges						Whitens. Ca flame.
Dolomite (pinkish curved crystals), Lockport.			+				yel-brown		Decrepitates. Turns brown. Ca. flame.
Dolomite (white), Pleasantville, N. Y.			+				yellowish		Turns yellowish. Ca. flame.
Ankerite, Brazil.			+					+	Decrepitates. Blackens. Ca. flame.
Ankerite, Erzberg, Styria.	+							+	Decrepitates. Blackens. Ca. flame. Slagged with forceps.
Magnesite (Breunnerite).			on edges					+	Blackens.
Magnesite, Greece.			+			+			Red flame noted.
Magnesite, Styria.			on edges						Red flame noted.
Rhodochrosite, Colo.	+							+	Decrepitates. Blackens.
Aragonite (crystal), Eisenerz, Styria.			on edges						Decrepitates. Whitens. Ca. flame.
Aragonite (crystal), Bilin.									Falls to pieces. No fusion test possible
Leucite, Rome.	+				+				Glass contains bubbles. K. flame.
Chrysolite, Hawaii.	+							+	Bead somewhat crystalline.
Chrysolite, Jackson Co., N.C.	+							+	Bead somewhat crystalline.
Zircon (Hyacinth), Expailly.			+			+			Loses color and whitens.
Zircon, Buncombe Co., N. C.			+			+			Loses color and whitens.
Zircon, (dark colored) Ceylon.			+			+			Loses color and whitens.
Dioptase, French Congo.	+						brown		Decrepitates. Turns brown. Cu. flame.
Chrysocolla.	+						brown		Decrepitates. Turns brown. Cu. flame.
Thorite, Arendal.	+						dk. br'n		
Topaz (yellow), Brazil.	+					+			Whitens. White vol. coat on forceps.
Topaz, Zacatecas.	+					+			White volatile coat on forceps.

FUSION TABLE.—*Continued.*

(A sign + indicates the tests obtained.)

Mineral.	Fusion.			Not Fused.	Bead or Glass Produced.				Remarks.
	Easy.	Fairly Easy.	With Difficulty.		Clear and Colorless.	White.	Colored.	Black.	
Andalusite (pink), Westford, Mass.	+					+			Loses color.
Sillimanite (crystals), Chesater, Conn.	+					+			Loses color.
Sillimanite, Newcastle Co., Del.	+					+			
Sillimanite, Bodenmais.	+					+			Loses color. Bead showed Fe. reaction.
Cyanite (blue).	+					+			
Chondrodite, Warwick.	+							+	Greenish flame noted.
Chondrodite, Tilly Foster, N. Y.									Violent decrepitation and scintillation. No fusion test possible.
Staurolite, St. Gothard.	+							+	
Kaolinite (white, mealy), Mitchell Co., N. C.	+				+				
Monazite, Burke Co., N.C.	+						honey-br.		Yellow-green flame.
Columbite, Chesterfield.	v. easy							+	Scintillates. Bead is crystalline, only slightly magnetic.
Wavellite (green), Ark.	+				+				Loses color, yellow-green flame.
Wavellite (white), York Co., Pa.	+				+				Yellow-green flame.
Turquoise, New Mexico	+							+	Decrepitates. Loses color, green fl.
Spinel (ruby crystal), Ceylon.		+					reddish		Does not change color.
Franklinite, Stirling Hill, N. J.	v. easy							+	Scintillates. Coating of Zn. oxide on forceps
Chromite, Lancaster, Pa.		+						+	Bead when crushed is magnetic.
Chrysoberyl, Brazil	+						dark gray		Bead is crystalline.
Chrysoberyl, Hadden, Conn.	+						yel.-white		Bead is crystalline.
Hausmanite, Thuringia.	v. easy							+	Scintillates.
Braunite, Thuringia.	v. easy							+	Scintillates. Bead is crytalline. Ba. flame noted.
Psilomelane, Westphalia.	+							+	
Wad, Wilson, Wis.	+							+	
Uraninite, Saxony.	+							+	
Aluminite, Sussex	+					+			Glass contain bubbles.

melting of any available holder or support. Instead of this method small mineral fragments (1–1½ mm. in thickness, about the size used for ordinary blowpipe fusion tests) were held in a pair of long, thin pointed, platinum-tipped forceps. With precaution there is very little danger of fusing or slagging the platinum tips, although special care should be employed in the treatment of the more easily fusible, ferruginous minerals. During these fusion tests many of the specimens, especially the lime and magnesia minerals, glowed very intensely and it was necessary to wear thick blue glasses and in addition at times to use dark glass screens. The high temperature also made it very possible to notice the characteristic flames.

During recent years many experiments on the melting points of minerals, have been carried on by Vogt,* Dœlter,† Joly,‡ Cusack,§

* J. H. L. Vogt, "Über die Schmelzpunkt-Erniedrigung der Silikat-Schmelzlösungen, u. s. w.," 1904.

Brun * and Tammann.† The high temperatures have been obtained in gas or electric furnaces, but so far as is known, up to the present time, no records have been made of simple fusion tests made with the high temperature of the oxy-gas blowpipe.

In the preceding table the common minerals recorded as "infusible" in the ordinary blowpipe flame, have been arranged in the systematic order of Dana and when deemed necessary the localities recorded, so as urnish a clue to the probable composition of the material used.

An examination of these results will show that in the case of the black minerals with metallic luster (iron and manganese minerals chiefly), the fusion effects are all about the same, namely, easy fusion to a black bead. Therefore no new practical scheme for determination is suggested. The minerals with non-metallic luster, however, show in many cases well marked differences, such as the ease of fusion and the color of the resulting bead. Some very convenient distinctions are here brought out. Quartz fuses to a clear bead, while topaz fuses to a white bead with the formation on the platinum forceps of a white volatile deposit of fluoride of silicon. Massive rutile and cassiterite both fuse easily to black beads, but the volatile white coat of tin oxide appears very quickly on the cassiterite and also on the forceps' tips. This coat can be turned blue-green with nitrate of cobalt. The dark colored, massive zircon fuses fairly easily to a white bead. Chrysoberyl fuses easily to a markedly crystalline, light colored globule. In the carbonate group the presence of iron or manganese causes easy fusion to black beads. The relatively easier fusion of massive magnesite over brucite can probably be accounted for by the admixture of sepiolite.

DEPARTMENT OF MINERALOGY,
COLUMBIA UNIVERSITY, NEW YORK CITY,
January 1, 1908.

†C. Dœlter, "Über die Bestimmung der Schmelzpunkte, u. s. w.," *Tscherm. Min. Petrog. Mitt.*, B. 20, 1901; 21, 1902; 22, 1903.

‡J. Joly, "On the Determination of the Melting Points of the Minerals," *Proc. Royal Irish Acad.*, Series III., Vol. 4, 1896.

§R. Cusack, "On the Melting Points of the Minerals," *Proc. Royal Irish Acad.*, Series III., Vol. 4, 1896.

*A. Brun, "Étude sur le Point de Fusions des Minéraux," *Arch. d. Sc. phy. et nat. de Genève*, Ser. IV., T. 13, 1902.

†G. Tammann, "Kristallisieren und Schmelzen.," 1903.

ABSTRACTS—ANALYTICAL CHEMISTRY.

By E. WALLER.

Sodium Carbonate and Zinc Oxide for determinations of sulphur or arsenic. Ebaugh and Sprague (*J. Am. Chem. Soc.*, XXIX., 1475). The reagent is made by thoroughly mixing one part of dry Na_2CO_3 with four parts of ZnO. 0.5 gm. of the material to be analyzed is placed in a porcelain dish and covered with an amount of the reagent sufficient to afford at least twice as much Na_2CO_3 as would be necessary to combine with the S or As present. The dish is then heated to redness in a muffle for 15 or 20 minutes. After cooling, the mass is extracted with water, affording a solution in which S or As may be determined. For S the detail would be as in the Eschka method; for As, it would be as in the Pearce method.

Decomposing Ores — Silica and Alumina in Iron Ores. Dean (*J. Am. Chem. Soc.*, XXIX., 1208). In a former paper the author recorded that Al_2O_3 adhering to SiO_2 after evaporation to dryness, could be rendered soluble in HCl by a short and not too strong ignition (vid QUARTERLY, XXVIII., 493). Further experiments along this line have shown that an ore can be directly ignited, and the bases then dissolved out by HCl; that ignition in a porcelain crucible is preferable, since overheating may thus be avoided; also that the use of a reducing agent (as $SnCl_2$) in the HCl facilitates the solution.

For SiO_2 only, take 0.5 gm. pulverized ore (100 mesh) in a 20 c.c. porcelain crucible, cover, and heat to dull red, then raise the heat for a few seconds, then remove from the heat, cool — transfer to a beaker — add 20 c.c. conc. HCl containing 1 gm. $SnCl_2$ in 225 c.c. Boil vigorously for some minutes after solution appears complete — altogether about 20 to 30 minutes — dilute and filter.

For SiO_2 and Al_2O_3 use 0.5 gm. in a similar crucible, place on top of the ore 0.25 to 0.4 gm. S, cover, heat up gradually to dull red, then raise the temperature for a few seconds, transfer to a beaker, pulverize with a glass rod, add 25 c.c. conc. HCl, and boil 15 minutes. Then add HNO_3 to clear the solution, and boil again for 5 or 10 minutes. A second addition of HNO_3 may be required, but its use must be avoided before the HCl has had the opportunity to attack the ore in presence of the reducing sulphide. The SiO_2 remains insoluble as before, and in the filtrate the Al_2O_3 may be determined.

Free Acid in Copper and Other Metallic Salts. Sims (*Chem. News*, XCV., 253). Half normal $NaHCO_3$ is used, the metallic bicarbonate being sufficiently soluble to give no interference by precipitation. Methyl orange is used as indicator. When in suitable proportion the reddish color of the methyl orange, and the blue of a copper solution, are com-

plementary and the solution is practically colorless. Excess of acid gives a pink, excess of alkali, a green tinge. 0.1 c.c. of N/2 $NaHCO_3$ or of N/2 H_2SO_4 will restore the colorless condition. Hence to conduct the process, dissolve 3 gms. of the copper sulphate crystals or take an amount of the material (liquor, etc.) in 100 c.c. in a 10 oz. Erlenmeyer flask. Add four drops of methyl orange solution. If the color is pink, titrate with N/2 $NaHCO_3$ to the colorless stage. The free acid present is then calculated from the number of c.c. used plus 0.1 c.c. If the color is green, the acidity is not over the equivalent of 0.1 c.c. $NaHCO_3$. In this case titrate to colorless with N/2 $H2SO_4$. The amount of free acid present is equivalent to 0.1 c.c. minus the number of c.c. of N/2 acid used. The method is also applicable to salts of nickel or iron.

Alkalies. Separation from Magnesium. Browning and Drushel (*Am. Jour. Sci.*, XXIII., 293). The Mg is separated by addition of 40 to 80 per cent. excess of $(NH_4)_2HAsO_4$ in a solution distinctly but not strongly ammoniacal. Complete precipitation is effected quickly either by freezing the solution in a mixture of ice and salt, or by addition of 15 to 20 per cent. of its volume of alcohol. In the filtrate the As can be completely removed by evaporation in platinum dish after addition of about 10 c.c. HCl (Gr. 1.2) and 10 c.c. HBr (Gr. 1.3) or 1 to 3 gms. NH_4Br. After ignition to remove NH_4 salts the alkaline chlorides can be converted to sulphates for weighing.

Magnesium in Waters. Pfeiffer's method. Cochenhausen (*Zts. f. angew. Chem.*, XIX., 1987 and 2023). The method consists in adding an excess of solution of $Ca(OH_2)$ of known strength, filtering and titrating the excess of $Ca(OH)_2$ with N/10 acid. Several sources of error may exist. They may, however, be to a great extent nullified or compensated by working in the following manner: Bring 200 c.c. of the water to a boil, then add a slight excess of $Ca(OH)_2$ solution (determined by a preliminary trial), cool, filter, dilute to 500 c.c. and titrate. By this means fairly satisfactory results can be obtained.

Magnesium in Waters — Volumetric. Frankforter and Cohen (*J. Am. Chem. Soc.*, XXIX., 1464). The method is a modification of that of Meade (*ib.*, 1899, p. 146) — precipitating with Na_2HAsO_4 and determining As in the precipitate by KI and standard $Na_2S_2O_3$ solution. The authors' procedure is as follows: Use 500 c.c. of the water to be examined; separate Fe, Ca, etc., by the usual methods, acidify and evaporate until the salts begin to crystallize out, make up to 100 c.c.; transfer to an Erlenmeyer flask, add one third its volume of strong NH_4OH, and 25 c.c. of a 10 per cent. solution of Na_2HAsO_4, cork and shake vigorously for 10 minutes, settle, filter, wash with as little dilute NH_4OH as possible, dissolve into the flask with 50 c.c. H_2SO_4 (1:4). Rinse down the paper with hot water until the solution measures about 100 c.c. add 10 c.c. H_2SO_4 (1:1) cool, and add 3.5 gms. pure KI allow to stand for 5 minutes and then titrate with standard $Na_2S_2O_3$ solution, *avoiding the use of starch indicator.* The strongly acid solution has an influence on the starch, which prevents a clear and trustworthy end-reaction. The yellow tint of the dissolved I which entirely dis-

appears when the end-point is reached, serves as a much more certain guide.

Electrolytic Separations. McCutcheon (*J. Am. Chem. Soc.*, XXIX., 1445). Experiments were made with the Hildebrand cell (*ib.*, 4, 447) in which a layer of Hg at the bottom of the dish forms the cathode, above which, in the liquid electrolyzed, is suspended a rotating anode of Pt gauze dipped in an inner cell made of a cut-off beaker, the lower edge of which beaker dips below the surface of the Hg. The Pt gauze was plated with Ag, and afterward with different metals. The results indicated that on the whole Ag was the most suitable anion in most cases. Salts of the alkalies and alkaline earths gave up their metals to form amalgams with the Hg, which decomposed in the outer cell, affording solutions which could be alkalimetrically titrated, while the chlorine or other electro-negative ion, combined with the Ag of the anode, making it possible in the case of chlorides and of a few other salts, to determine both constituents of a salt by one treatment. In the case of a mixture of $BaCl_2$ with $CaCl_2$ and $MgCl_2$ with a pressure of 3.5 volts, only Ba appeared in the outer cell, and could be accurately determined. By increasing the pressure to 8 volts the Ca could also be separated. Separations of Ba from U and from cerite metals were also successfully accomplished.

Chromium Alloys. Galli (*Atti R. Accad. Lincei*, XVI., 58). The most convenient method of dissolving ferrochromes and alloys of that character consists in suspending the alloy in a 15 per cent. solution of KCl rendered slightly alkaline with KOH, attaching the alloy to the negative pole of a battery, and using an E.M.F. of 8 to 10 volts and 0.5 amp., the temperature of the solution being kept at 80° to 85°. The pole should be surrounded by a porous pot to prevent reduction of K_2CrO_4 by the H evolved. The final result is a solution of K_2CrO_4, with a precipitate of $Fe_2(OH)_6$.

Metallic Iron in Presence of its Oxides. Martin (*J. Am. Chem. Soc.*, XXIX., 1211). The method consists in dissolving in acid, and measuring the H evolved. An illustration and description of the apparatus is given.

Iron Analysis. Report of Committee on Standard Methods. (*J. Am. Chem. Soc.*, XXIX., 1372.)

Silicon. Dissolve 1 gm. in 30 c.c. HNO_3 (Gr. 1.13) and 5 c.c. conc. H_2SO_4. Evaporate to disappearance of fumes (?). Take up with water, boil to dissolve Fe salt. Filter, wash once with hot water, once with HCl, then three or four times with hot water. Ignite and weigh. Expel SiO_2 by treatment with HF and H_2SO_4 and weigh again. Loss $\times$ 0.4702 = Si.

Sulphur. Dissolve 3 gms. in conc. HNO_3 in a covered platinum dish. Then add 2 gms. KNO_3, evaporate to dryness and ignite overan alcohol lamp. Add 50 c.c. of a 1 per cent. solution of Na_2CO_3, boil and filter, wash with hot Na_2CO_3 solution, acidify the filtrate with HCl, evaporate to dryness, take up with 50 c.c. water and 2 c.c. conc. HCl, filter, wash, dilute to 100 c.c. and precipitate with $BaCl_2$.

Phosphorus. Dissolve 2 gms. in 50 c.c. HNO_3 (Gr. 1.13) add 10 c.c. HCl, evaporate to dryness, and bake off the free acid. Redissolve in 25 to 30 c.c. HCl and add an equal bulk of water. Filter and wash. Evaporate to about 25 c.c., add 20 c.c. conc. HNO_3, evaporate to a film, add 30 c.c. HNO_3 (Gr. 1.2) and evaporate again to a film. Dilute to 150 c.c. with hot water. When the temperature of the mixture has come down to 70° or 80° C., add 50 c.c. of molybdate solution, agitate for a few minutes and filter through a weighed Gooch. Wash three times with 3 per cent. HNO_3 and then twice with alcohol. Dry at 100° to 105° and weigh the yellow precipitate. Multiply by 0.0163 to get P.

For *manganese* the sodium bismuthate method is prescribed.

For the *carbon* combustion the use of a preheater, or of the Shimer crucible is made optional.

The methods given are not so much for rapidity as to afford check methods in case of dispute between laboratories, etc.

Titrating Iron in Presence of Titanium. Gooch and Newton (*Am. J. Sci.*, XXVIII., 365). On reducing a solution from a titaniferous ore or mineral with Zn, TiO_2 is reduced as well as the Fe_2O_3. The authors find that by the addition of a little $CuSO_4$, or CuO, or better of Bi_2O_3, the Ti is reoxidized without affecting the FeO. By passing the solution through a rapid filter, the permanganate titration of the filtrate gives only the Fe.

Titanium-Colorimetric. Walton (*J. Am. Chem. Soc.*, XXIX., 481). Fuse a few decigrammes of the mineral with 4 to 5 gms. Na_2O_2 in an iron or nickel crucible. With ferruginous clays, use 8 gms. Na_2O_2 to 1 gm. of the sample. Dissolve in 300 c.c. of *cold* water, filter and acidify with H_2SO_4. The result is a coloration due to the same Ti compound as is afforded by addition of H_2O_2 to a solution of TiO_2, which, however, is destroyed by heating, hence the necessity of keeping the solutions cold. In presence of much Fe some Ti was retained in the insoluble portion. On bringing this into solution with 15 c.c. dilute H_2SO_4 (1 : 1) and adding 6 c.c. of 50 per cent. solution of H_3PO_4, and adding H_3PO_4 also to the standard, correct color comparisons could be effected.

Manganese in Tungsten Steels. v. Knorre (*Stahl u. Eisen*, XXVII., 380). In the absence of O, H_2SO_4 does not dissolve the W. The solution is therefore effected in an atmosphere of CO_2. A sufficient amount of dilute H_2SO_4 is placed in a flask, and some of a saturated solution of $NaHCO_3$ is added so as to fill the flask with CO_2. A weighed amount of the steel is then dropped in, and the flask closed by a cork carrying a tube bent twice at right angles. The free end of the tube is dipped below the surface of a solution of $NaHCO_3$, and then on warming the steel is decomposed. After this, boiling for a few minutes completes the solution of the Mn. On cooling the solution is filtered, and in the clear filtrate the Mn may be determined, the W being left in the insoluble residue.

Manganese in Waters. Weston (*J. Am. Chem. Soc.*, XXIX., 1074). Evaporate enough of the sample to give 0.01 to 1 mgm. Mn, with 25 c.c. HNO_3 (Gr. 1.135). Ignite gently, or bake ½ hour at 130°, add 50

c.c. HNO_3 (same strength), cool, and add about 0.5 gm $NaBiO_3$. Heat until colorless. If any insoluble MnO_2 appears at this point, add a drop or two of dilute solution of $Na_2S_2O_3$ and heat. In any case cool the clear solution, and then add $NaBiO_3$ in excess, then filter through a thoroughly washed felt of asbestos in a Gooch crucible, wash with dilute HNO_3 (30 c.c. per liter) running the whole into a Nessler tube, bringing the volume up to 100 c.c. In another Nessler tube place 100 c.c. diluted H_2SO_4 (25 c.c. conc. acid per liter) and into this run standard $K_2Mn_2O_8$ solution to match the color. This latter is made up with 0.288 gm. $K_2Mn_2O_8$ per liter 1 c.c. = 0.001 gm. Mn.

Tungsten in Steel. Lind and Trueblood (*J. Am. Chem. Soc.*, XXIX., 477). Dissolve 2 to 10 gms. of the borings in about 30 c.c. HNO_3 (Gr. 1.2), adding 1 or 2 gms. $KClO_3$ if necessary. Evaporate to dryness, take up with 15 to 20 c.c. conc. HCl, evaporate low, dilute to 150 c.c., boil a short time and settle. Wash by decantation with dilute HCl (5 to 10 per cent.) until free from Fe. Wash free from acid with 3 to 5 per cent. solution of $NaNO_3$. Treat residue and filter paper with excess of N/5 KOH, heating if necessary. Cool and titrate back with N/5 HCl, using phenolphthalein indicator.

Nickel Qualitative. Pozzi-Escott (*Compt. Rend.*, CXLV., 435). $NiMoO_4$ was found to be insoluble in solutions containing excess of alkaline molybdate, even if weakly acid. The corresponding Co compound is soluble. The solution to be tested is made nearly neutral by addition of alkali, and then an excess of aqueous solution of $(NH_4)_2MoO_4$ is added. On heating to 70° C. and agitating, a crystalline greenish white precipitate forms if Ni is present. If Co is present the solution assumes a rose color.

Nickel, New Method, and Separation from Cobalt or Zinc. Grossman and Schück. (*Zts. Angew. Chem.*, XX., 1642). By using dicyandiamidin salts as the sulphate, dicyan-diamidin nickel $(C_2H_5N_4O)_2Ni$, 2 aq., is separated in an alkaline solution. The method is managed thus: The concentrated solution of the Ni salt is rendered strongly ammoniacal, and 4 to 5 times as much dicyan-diamidin sulphate also in concentrated solution is added, and then a 10 per cent. KOH solution is added slowly with constant stirring, until a point is reached when the color of the solution suddenly changes from blue to yellow. A crystalline yellow precipitate then begins to separate. Standing over night is recommended, though the separation is complete in less than six hours. The presence of large amounts of NH_3 salts is to be avoided, as they have a solvent effect. Filter and wash with ammoniacal water. If filtered through a Gooch, the precipitate may be dried and weighed as such. If not, it may be separated from the paper (which is incinerated separately) and the Ni converted to $NiSO_4$ for weighing. In presence of Co first add a little NH_4Cl, and then NH_4OH in excess, then a few drops of H_2O_2 solution, and allow to stand for half an hour before proceeding as above directed. For separation from Zn the treatment is the same except that the use of H_2O_2 is unnecessary. Separation from Fe_2O_3 or Al_2O_3 can also be effected if alkaline tartrate is added to the solution.

Nickel in Steel. Determination in presence of Cr, Fe, Mn, etc.

Johnson (*J. Am. Chem. Soc.*, XXIX., 1201). The method is a modification of that of T. Moore (*Ch. News*, LXXII., 92, vid. Ibbotson and Brearley, p. 183) — KCN titration after addition of citric acid. With moderate amounts of citric acid, the deep color of the solution renders observation of the end point exceedingly difficult, and when Cr is present this difficulty is greatly increased. The larger the proportion of citric acid, the less the color. The following method, when closely followed was found to give satisfactory results: Dissolve 1 gm. of the steel in a small (No. 2) beaker with 20 c.c. HCl (1:1). When action ceases add 10 c.c. HNO_3 (Gr. 1.20). Boil down to 15 c.c., keeping the beaker covered. Add 8 c.c. conc. H_2SO_4 diluted with 24 c.c. H_2O. Transfer the whole to a beaker of 600 c.c. capacity containing 12 gms. citric acid (if Cr is present, use double this amount of citric acid). Stir until the acid is dissolved. Add diluted NH_4OH (1:1) until the solution is *faintly* but *distinctly* alkaline. The color changes can here be made of service in controlling the progress of neutralization. The first change is to amber, then yellowish green, then green, which becomes paler passing to a yellow, at which stage the neutralization is nearly complete. Excess of NH_4OH causes low results. A slight excess gives a brownish shade. Cool the mixture completely. Then add 2 c.c. of a 20 per cent. solution of KI and also run in from a burette, a measured volume of standard $AgNO_3$ solution. Then titrate with standard KCy solution until the cloud of AgI just disappears by being dissolved by excess of KCy. From the burette reading deduct that due to the $AgNO_3$ added, and calculate the Ni present. Solutions required, and standardizing:

KCy 4.4868 gms. in one liter. 1 c.c. = about 0.001014 gm. Ni.

$AgNO_3$ 2.925 gms. in 500 c.c.

To standardize use 1 gm. of a plain steel to which some known amount of Ni has been added. A convenient form is the C.P. $(NH_4)_2Ni(SO_4)_2 6H_2O$. After carrying through the titration in the manner described above, using 1.5 to 2 c.c. of the standard $AgNO_3$, add 10 of the KCy solution, and titrate with standard $AgNO_3$ until a distinct cloudiness is obtained. The result indicates the relation between the two solutions. The aim is to have them equal c.c. for c.c.

If a stronger KCy solution is used, the process may be applied to alloys containing larger proportions of Ni than are found in steels. A number of experiments are quoted, showing that Fe, Cr, Mn, Vd, W, Mo, etc., do not interfere with accuracy. Cu is of course well known to cause inaccuracy, but the author regards it as of infrequent occurrence in the samples usually encountered.

Nickel in Steel—Rapid Method. Dougherty (*Iron Age*, April, 1907, through *Chem. News*, XCV., 261 and 268). The method consists in dissolving in HNO_3 precipitating the Fe by NH_4OH in presence of a large amount of NH_4Cl, with special manipulation, and in the filtrate titrating with standard KCy.

The solutions required are: NH_4Cl solution 555 gms. in 2 liters; filter if necessary.

Standard. Ni solution 2.004 gms. Merck's "Highest purity" nickel dissolved in HNO_3 and diluted to 2 liters. Orford Copper Co.'s commercial nickel may be used, rating it at 99.25 per cent. pure.

Standard KCy, 24 gms. in 2 liters. When freshly made 1 c.c. should equal about 0.0025 gm. Ni. It should be standardized every few days using 50 c.c. of the Ni solution each time.

The author found that the amount of Ni usually occurring in steels could be completely separated from the Fe when sufficient excess of NH_4Cl was used.

The method is: "Dissolve 1 gm. of the steel in 15 c.c. HNO_3 (Gr. 1.20) and boil one minute. Pour the solution and rinsings into 270 c.c. of the hot solution of NH_4Cl. Add HCl if necessary to clear up, then NH_4OH little by little until the solution, though practically free from precipitate, is too strongly colored to see through in a good light. Cool quickly, transfer to a 500 c.c. flask, add all at once 50 c.c. NH_4OH, whirl, and fill up to the mark. Mix by shaking, and pour through a 10-inch filter into a clean dry flask. Take 250 c.c. of the filtrate ($\frac{1}{2}$ gm. of sample) render just acid with HCl, then cautiously add NH_4OH in very slight excess. Not more than 1 c.c. NH_4OH in excess is allowable. Cool, add 5 c.c. $AgNO_3$ solution (0.10 gm. in 200 c.c.) and then 5 c.c. of a 2 per cent. solution of KI. Stir and titrate with standard KCy until the last drop destroys the white opacity of the liquid. Deduct 0.2 c.c. from the burette reading, and then calculate proportion of Ni.

If the steel contains Cu dissolve 1 gm. in 20 c.c. HCl (Gr. 1.10). Add $2\frac{1}{2}$ c.c. HNO_3 and boil one minute. Proceed as previously described as to separation of Fe. In the 250 c.c. which, in the absence of Cu would be directly titrated for Ni, just acidify with HCl, then add 14 c.c. more HCl to bring the amount of free HCl to 5 per cent.; pass H_2S to separate Cu, filter, boil out H_2S, neutralize in the manner already described, and titrate.

Zinc Oxide in Zinc White, etc. Tambon (*Bull. Soc. Chim.* [4], I., 823). Prepare a mixture of equal parts of:

Ammonia (Sp. Gr. 0.924),
Ammonium carbonate 20 per cent. solution,
Ammonium chloride 20 per cent. solution.

In this ZnO is soluble even after ignition.

In the case of Zinc white shake 10 gms. of the sample with 300 c.c. of the solution a short time, let stand, filter and weigh up insoluble material or determine Zn volumetrically in the solution. With "Zinc gray" or lithopone allow to digest for 30 minutes.

A preliminary washing with water to remove soluble Zn salts may be necessary with some pigments before applying the treatment.

Mercury by Electrolytic Methods. Borelli (*Gazetta*, XXXVII., 425). Drying the Hg for weighing after electro-deposition is accomplished with difficulty without loss of Hg. The author finds that it is best accomplished by placing it in a desiccator containing fused KOH for four or five hours. A small capsule containing Hg may also be placed in the desiccator to prevent any loss by volatilization.

Volumetric for Mercury. Robertson (*Chem. News*, XCV., 253). The Hg must be in *ic* form (produced if necessary by boiling with

HNO_3) and chlorides must be absent. The amount of material used should not contain over 0.04 gm. Hg. To the solution is added 10 c.c. of N/10 NH_4CNS (1 c.c. = 0.005 gm. Hg) and then excess of $ZnSO_4$ solution. A precipitate of $ZnHg(CNS)_4$ separates which is filtered off and washed thoroughly with dilute $ZnSO_4$ solution. The precipitate is only soluble to the extent of 2.4 gms. in 1,000 c.c. of water, and is still less soluble in presence of Zn salts. In the filtrate the excess of NH_4CNS is determined by adding 10 c.c. N/10 $AgNO_3$ and titrating back with N/10 NH_4CNS with $Fe_2(SO_4)_3$ indicator.

For small amounts of Hg, it may be convenient to use volumetric solutions of half the above strength, but it is advisable not to work at too great dilutions. Excess of HNO_3 does not interfere.

Mercurous Hg — Titration. Randall (*Am. J. Sci.*, XXIII., 137). Direct titration with standard permanganate alone is impossible, as the solution turns brown from the higher oxides of Mn, and the end-point is uncertain. But by adding an excess of N/20 permanganate until the solution is distinctly red, and titrating back with N/20 $FeSO_4$ solution, an accurate determination can be made. The presence of free HNO_3 up to 3 per cent. was found not to interfere. The lower N oxides will cause error.

Small Amounts of Arsenic. Sanger and Black (*J. S. C. I.*, XXVI., 1115). The authors have worked out a quantitative method with use of the Gutzeit test, a paper impregnated with $HgCl_2$ being wholly inserted in the delivery tube of the apparatus (a simple form of which is described and figured) through which the AsH_3 escapes. The paper strips are made uniformly 4 mm. wide and 7 cm. long. A set of standards for comparison are made up using known amounts of As_2O_3 (from 0.02 to 0.7 mgms. As_2O_3). The color band is more markedly developed by treatment of the paper slip (after making the test) with HCl (1:1) for not over two minutes, washing and drying and then sealing up in a tube containing a little P_2O_5 under a cotton-wool plug.

For evolution of the H, HCl (1:6) is preferred. For details the original paper should be consulted.

Tin from Arsenic — Electrolytic. Lampen (*Chem. Ind.*, XXX., 128). With a strongly alkaline solution (3 to 3.5 gms. KOH to 0.1 to 0.2 gm. Sn) and a temperature of about 80 to 85° C. Sn can be completely separated from As with a current of 0.6 amp. per 100 $cm.^2$ and 3.6 to 4.2 volts. This Sn should be mostly present in stannic form. After half an hour the current density should be increased to 0.75 amp. and the temperature slightly raised. Three or four hours are required for complete separation. NH_4 salts should not be present.

Tin in Tin Stone. Koomans (*Jaarb. Mijnwesen Nederl. Oost. Ind.* XXXX., 171). Pulverize finely, mix 1 to 1.5 gms. with 20 times its weight of Zn powder, and treat the mixture with enough 16 per cent. HCl to dissolve about 0.9 of the Zn. After the action is over decant off as closely as conditions permit, and add excess of HCl, to dissolve Sn sponge and surplus Zn. Then filter, dilute to 100 c.c., heat to boiling and insert a rod of Zn. The deposition of Sn in form of a sponge which can readily be detached from the Zn rod, usually is complete in

about 16 minutes. After washing, this is converted to SnO_2 by the aid of HNO_3 and weighed in that form.

Osmium. Paal and Amberger (*Ber.* XL., 1378). Examination of methods proposed for quantitative estimation. Precipitation by NH_4Cl was found to be incorrect. Also separation by Al, whether amalgamated or not. In general terms, the precipitates brought down by reducing agents are not entirely metallic Os but contain some OsO_2 or its hydrate. Warming OsO_4 for a time with a little KOH (3 c.c. of 15 per cent. solution) to form K_2OsO_5 and then adding a few c.c. of alcohol and acidifying with dilute H_2SO_4 and allowing to stand for 10 to 12 hours, gave a blackish precipitate, which when reduced in a current of H at 230° to 250° was converted into metallic Os, in a form suitable for weighing. The results were accurate. Correct results were also obtained by reducing alcoholic solutions with aqueous solution of formaldehyd. This produces at first a colloidal solution, but by standing the precipitate gradually forms. It requires reduction in H as before, to convert it entirely to metallic form.

Lanthanum — Volumetric. Drushel (*Am. J. Sci.* [4], XXIV., 197). The La is precipitated as oxalate, the precipitate after thorough washing being dissolved in H_2SO_4(1:4) and the solution then titrated with standard permanganate. If a known amount of a standard solution of $H_2C_2O_4$ or $(NH_4)_2C_2O_4$ is used as the precipitant, the La may be indirectly determined by titrating the filtrate.

Separation of Chloride and Bromide. Andrews (*J. Am. Chem. Soc.*, XXIX., 275). The direct determination of chloride in crude alkaline bromide is effected by boiling with KIO_3 and dilute HNO_3, which expels the Br. The excess of HIO_3 is then destroyed by phosphorous acid, and in the residual solution the Cl is determined by the Volhard method (standard $AgNO_3$ and standard NH_4CNS). For details the original paper should be consulted.

As regards Cl in crude Br, the author finds that even pouring the Br from one bottle to another will cause a loss of Cl, which may be present in as large proportion as 13 per cent. or more.

A specially constructed pipette is described for taking out the samples for weighing. The mode of analysis recommended consists in shaking up a weighed amount of the Br with KI solution, and then treating the resulting solution in the manner above suggested.

Chlorides or Bromides in Presence of Sulphocyanates. Rosanoff and Hill (*J. Am. Chem. Soc.*, XXXIX., 1467). It was found possible to destroy the sulphocyanate by boiling with HNO_3 and then to obtain a solution in which the Cl (or Br) could be determined by the Volhard method. The volumetric solutions of NH_4CNS and $AgNO_3$ used were fifteenth normal, the indicator solution a saturated one of iron ammonium alum containing some free HNO_3. To destroy the CNS ion, the solution is diluted to about 50th normal with respect to the chloride present (100 normal in the case of Br). This is brought to a gentle boil (1 c.c. of the indicator solution being added) and conc. HNO_3 is added at the rate of about 3 drops per minute, stirring frequently to prevent boiling over. When the color of the solution

becomes a *pale orange*, the addition of acid is stopped, and the boiling is continued for about 5 minutes. Cool the solution and add one third its volume of conc. HNO_3 (one sixth in the case of Br). Then add a measured amount of N/15 $AgNO_3$ in moderate excess over what is necessary to combine with the Cl (or Br) present and agitate well. In a cool solution the precipitate coagulates quickly but *not when heated.* When well coagulated, boil for 5 minutes and filter. (Filtration is omitted in the case of Br.) No attention needs to be paid to the opalescence developing as the filtrate cools. Wash with hot water acidified with HNO_3. Cool, dilute to about three times the volume before filtration, add 2 c.c. of indicator, and titrate until a single drop of the NH_4CNS, gives a pink tinge, removed by one drop of the $AgNO_3$ solution. If the point is overrun, run in a little from the $AgNO_3$ burette, and try again.

The total c.c. of $AgNO_3$ used minus total NH_4CNS used, is the $AgNO_3$ equivalent of the chloride (or bromide) present.

Iodide in Presence of Chloride and Bromide. Riegler (*Fres. Zts. Anal. Chem.*, XXXVI, 315). In alkaline solution, $K_2Mn_2O_8$ converts iodides into iodate, thus:

$$KI + 2KMnO_4 + H_2O = KIO_3 + 2KOH + 2MnO_2.$$

Then addition of hydrazin sulphate causes evolution of N which may be measured;

$$2KIO_3 + 3N_2H_4.H_2SO_4 = K_2SO_4 + 2H_2SO_4 + 2HI + 6H_2O + 3N_2.$$

Not more than would correspond to 0.2 KI should be used. The solution should be about 40 c.c. in bulk. Heat to boiling, add a few crystals of $K_2Mn_2O_8$ and keep up the boiling with frequent agitation of the solution. When an intensely red color is permanent after boiling 1 to 2 minutes the conversion to iodate is complete. Remove from the heat, add a pinch of glucose to destroy the surplus $K_2Mn_2O_8$.

Filter off the MnO_2 and wash, bringing the whole volume to about 60 c.c. in a 150 c.c. flask, add diluted H_2SO_4 (1:5) until the reaction is decidedly acid. Stopper the flask and shake well, open up to allow CO_2 to escape, and repeat this treatment several times to remove the CO_2. The solution is then brought in contact with 0.5 gm. hydrazin sulphate in an evolution flask, which is connected with a graduated burette. During the operation the evolution flask is kept immersed in water at the room temperature. Toward the end of the operation the flask must be taken out and shaken until the contents become completely colorless.

1 mg. N corresponds to 3.94 mgs. KI or 3.01 mgs. I.

Commercial Iodine. Cormimboeuf (*Ann. Chim. Anal.*, XII., 307). Treat 3 gms. of the sample with 25 c.c. H_2O, adding 1 gm. of iron filings. After thorough digestion, a pale green solution of FeI_2 is obtained, which is filtered, and the filtrate and washings are made up to 250 c.c. Of this solution, 50 c.c. are treated with 1 gm. Na_2CO_3 dissolved in a little water, and the whole is then diluted to 100 c.c. After mixing, this is filtered and 50 c.c. of the filtrate (= 0.3 gm.) are acidified with HNO_3, one fourth its bulk of NH_4OH solution added, and then $AgNO_3$. The AgI separated can then be determined, while in the ammoniacal filtrate Cl or Br may also be estimated.

Volumetric for Sulphuric Acid in Waters. Komarowsky (*Chem. Ztg.*, XXXI., 498). A modification of Andrews' method. The water is acidified with HCl, and heated to boiling. Some pure dry $BaCrO_4$ is is then added, and the solution allowed to cool. Then NH_4OH is added, and the mixed $BaSO_4$ and $BaCrO_4$ filtered off. In the filtrate the CrO_3 resulting from the interchange with sulphates is determined volumetrically by KI and standard $Na_2S_2O_3$ and from this the SO_3 present may be calculated.

Phosphoric Acid — Volumetric. Richardson (*J. Am. Chem. Soc.*, XXIX, 1314). In applying the Pemberton volumetric method in analyzing fertilizers, etc., it was found that the presence of H_2SO_4 or of sulphates invariably gave high results (0.2 to 1 per cent.).

An examination of the "yellow precipitate" produced under the above conditions showed that it always contained SO_3, leading to the supposition that some complex ammonium sulphomolybdate is formed.

Carbon Monoxide in Atmospheric Air. Morgan and McWhorter (*J. Am. Chem. Soc.*, XXIX., 159). All of the satisfactory methods are based on the reaction; $I_2O_5 + 5CO = 5CO_2 + I_2$ which occurs readily at 150° and above. The I thus set free is determined. The Kinnicutt and Sanford method (*J. Am. Chem. Soc.* XXII., 14) was used, which consists in absorbing in KI solution, and titrating with standard $Na_2S_2O_3$ solution. It was found hat the U tube containers of the I_2O_5 should be sealed off by the blow pipe after filling since the presence of organic matter, such as might come from the lubricant used with glass stoppers also causes decomposition of the I_2O_5.

By passing the gases (after removal of I) through a standard solution of $Ba(OH)_2$ and subsequently titrating with $H_2C_2O_4$, check results are easily obtained. Interposing another $Ba(OH)_2$ tube before passing the air over I_2O_5 affords a means of determining the CO_2 already in the air by the same operation.

Volumetric for Nitrates. Vriens (*Fres. Zts. Anal. Chem.* XLVI., 414). The oxidizing influence on a ferrous salt is utilized, the presence or absence of excess of ferrous salt being found by repeated trials. The solutions are:

1. 25 gms. Fe $(NH_4)_2(SO_4)_2.6H_2O$ (Mohr's salt) in one liter. This is standardized by the method described below. A little free H_2SO_4 is added.

2. 5 gms. pure $NaNO_3$ (or a corresponding amount of KNO_3 or NH_4NO_3) in one liter. (For standardizing.)

3. A 0.1 per cent. solution of $K_6Fe_2Cy_{12}$.

4. Concentrated H_2SO_4.

Suppose a Chili saltpeter is to be tested. Dissolve 5 gms. in a liter of water filter and take 10 c.c. of the solution at a time for the tests. To this, in a 200 c.c. flask, add about 10 c.c. of conc. H_2SO_4 and then a known amount, say 28 c.c., from a burette of solution 1. Boil violently over a free flame for two minutes with shaking, then add 1 c.c. of the ferricyanide solution. Half a minute after note the color. If some unoxidized FeO remains, the color will be blue; if none, brown. The aim is in the first trial to have ferrous in excess; in the second, to

have an insufficient amount, and in subsequent trials to approximate more closely; *e. g.*, on a second trial with 10 c.c. of the diluted nitrate solution (in case of standardizing) 27 c.c. of No. 1 gave a brown, though 28 gave a blue, 27.6 gave a blue-green, 27.4 a brown. The conclusion was that 27.5 showed the standard.

The method has been tested on mixtures of nitrates with K_2SO_4 $(NH_4)_2SO_4$ and $Na_2C_2O_4$ and has shown good results.

Nitrous Acid in Presence of Nitric. Moderna and Coffetti (*Gaz.*, XXXVII., 595). A nitrite in presence of free acetic, oxalic, tartaric or citric acids and K_4FeCy_6 affords NO thus:

$$K_4FeCy_6 + KNO_2 + 2HC_2H_3O_2 = K_3FeCy_6 + 2KC_2H_3O_2 + NO.$$

The solution when shaken with citric and K_4FeCy_6 in a Lunge nitrometer gives NO quantitatively, the volume of which makes a basis for calculation of the nitrite. HNO_3 causes no interference.

Organic Nitrogen in Water. Rubner (*Arch. Hyg.*, LXII., 83). The Kjeldahl process is used, 200 c.c. or more being boiled down in a flask after addition of 5 to 8 c.c. conc. H_2SO_4 and 1 gm. K_2SO_4, the material being treated otherwise as usual. Bumping may be prevented by adding a little talc. To determine the nitrogen in the suspended matter 5 to 20 liters of the water are precipitated with ferric acetate at a gentle heat, and the residue which contains all of the bacteria, etc., is treated by the Kjeldahl process.

Hydrogen in Mineral and Organic Substances. Lidoff. (*Fres. Zts. Anal. Chem.*, XLVI., 357). The substance is mixed with Mg in powder, the latter in liberal excess, and then ignited in a test-tube which is connected with a eudiometer, in which the H evolved is measured. When the percentage of H is under 3, use 200 to 500 mgs. of the substance; if 3 to 7 percent. use 100 mgs.; if 7 to 10 use 50 mgs.; if 10 to 15 or more use 25 mgs. Mix with 0.6 to 1 gm. Mg powder in a capsule, transfer to the test-tube rinsing the capsule with some Mg powder and if necessary add more Mg powder to bring the depth to 80 mm. Place a spiral of Al foil above the mass, test the tightness of the apparatus when connected up, and if that is satisfactory produce a slight rarefaction of the air within the tube, and then apply heat — at first to the layer of Mg powder above the mixture until that glows, eventually carrying it to the mixture itself.

With fat acids and nitrogenous substances (as acetamid, etc.) the errors were very considerable, but with NH_4Cl, various crystallized salts, cyanides, sugar, starch, etc., they were comparatively small.

Lithopone Analysis. Repiton (*Ann. Chim. anal. app.* XII., 93). To value the material, the author regards it only as necessary to determine the amounts of ZnS and ZnO. To determine the first, a weighed quantity of the material is titrated in a flask with HCl, and the evolved H_2S is absorbed in a standard ammoniacal solution of Cu. After filtering off the CuS the excess of Cu remaining is titrated with KCN. To determine ZnO, treat another weighed portion with $NaC_2H_3O_2$ containing a considerable excess of acetic, and in the solution the Zn is determined by any convenient method.

White Lead. Davis and Klein(*J. S. C. I.*, XXVI., 848). For

PbO, ignite 1.5 gms. in a porcelain crucible to constant weight. If the heat is made strong enough to partly fuse the PbO no red lead will be formed. For CO_2 dissolve 3 gms. in a slight excess of dilute HNO_3, and absorb evolved CO_2 in a potash bulb of the Wetzel type (vid. Davis, *J. S. C. I.*, XXV., 792). H_2O may be determined by heating in a combustion tube to above 200° C., passing a current of dry air, and absorbing in a weighed $CaCl_2$ tube.

When the pigment has been ground in oil it was found impossible to extract absolutely all oil with ether.

ABSTRACTS OF RECENT MINING DECISIONS.*

LOCATION OF CLAIMS.

PLACER MINES AND OIL CLAIMS.

The laws governing the location of placer claims on the public domain apply with equal force to the location of oil claims.

Woolfskill *v.* Smith, (Cal.) 89 Pac. 1001. (March, 1907.)

ABANDONED LANDS REVERTED TO THE GOVERNMENT.

Public land, upon which an oil company had drilled two artesian wells and which was then abandoned, reverted with such wells, to its original status as unoccupied and government land.

Woolfskill *v.* Smith, (Cal.) 89 Pac. 1001. (March, 1907.)

SUFFICIENCY OF LOCATION.

A location on the discovery of a mineral-bearing vein, made on land subject to location, was not rendered entirely void because corners were not placed on unappropriated land subject to location; but such location was valid to the extent that it was within the marked boundaries and on unappropriated land.

McElligott *v.* Krogh, (Cal.) 90 Pac. 823. (June, 1907.)

LOCATION — VOID AS TO EXCESS.

A mining location made in good faith but which included within its boundaries more than the statute permitted, being more than 300 feet, on each side of the middle of the vein at the surface, was held to be void only to the extent of the excess.

McElligott *v.* Krogh, (Cal.) 90 Pac. 823. (June, 1907.)

LOCATION — CORRECTION OF LINES AND CORNERS.

Where locators failed to place a monument at an intervening point on the line between the end monuments, as under their mistaken belief as to the accuracy of the location of the end monuments, there could be no necessity for it, they were entitled to have the line established on the correction of the location of one of the end monuments, which would include the corrected corner, that point, and the original corner not corrected, where no part of such line was more than 300 feet from middle of the vein; and on the correction of this corner, the court was not required to fix the boundary line as a straight line between the corner corrected and the original corner not corrected.

McElligott *v.* Krogh, (Cal.) 90 Pac. 823. (June, 1907.)

* Prepared for the SCHOOL OF MINES QUARTERLY by J. W. Thompson, Attorney at Law, Indianapolis, Indiana.

LOCATION — EXTRALATERAL RIGHTS.

Where the apex of a vein between two points was outside of the mining claim, and entirely within the boundaries of another, it was held that no extralateral rights in favor of the owner of such former claim cou d apply as to segment of the vein between such points.

McElligott *v.* Krogh, (Cal.) 90 Pac. 823. (June, 1907.)

MINING CLAIMS — PERSONS WHO MAY ACQUIRE AND LOCATE.

A government mineral surveyor appointed under the United States Statutes, Section 2334, was not an officer, clerk, or employee in the general land office, within the meaning of Section 452 of the United States Statutes prohibiting certain persons from obtaining title to government land; therefore, such government mineral surveyor was not disqualified from locating a mining claim under Section 2207 of the United States Statutes.

Hand *v.* Cook, (Nev.) 92 Pac. 3. (October, 1907.)

OIL LAND — LOCATION.

The failure of the locator to discover oil and a failure to retain possession and prosecute the work, was held fatal to the validity of the location under the statute providing that the entry of petroleum lands should be governed by the law relating to placer claims.

New England, etc., Oil Co. *v.* Congdon, (Cal.) 92 Pac. 180. (October, 1907.)

PLACER AND LODE LOCATIONS — DISCOVERY OF MINERAL.

The validity of both placer and lode locations depends on the discovery of mineral within the limits of a mining claim.

New England, etc., Oil Co. *v.* Congdon, (Cal.) 92 Pac. 180. (October, 1907.)

MINING LOCATION — POSSESSION — LACK OF DISCOVERY OF MINERAL.

The mere posting of notice and marking boundaries on the ground will not exclude others from a peaceable entry where the first locator had made no discovery and had not retained possession and prosecuted the work looking to a discovery.

New England, etc., Oil Co. *v.* Congdon, (Cal.) 92 Pac. 180. (October, 1907.)

MINING CLAIM — LOCATION — DISCOVERY.

It is not necessary that the discovery of mineral should precede the location of a mining claim in order to sustain the location.

New England, etc., Oil Co. *v.* Congdon, (Cal.) 92 Pac. 180. (October, 1907.)

LAND SUBJECT TO RELOCATION — PROOF.

Where an oil claim was located and the boundaries marked, but no discovery was made, and a cabin was constructed without furniture,

doors, or windows, and no one lived in it, and a watchman was employed to watch the claim in connection with other lands, but no drilling or actually developed work was done nor any steps taken in that direction, the land was held subject to relocation.

New England, etc., Oil Co. *v.* Congdon, (Cal.) 92 Pac. 180. (October, 1907.)

INSUFFICIENT DESCRIPTION OF CLAIM.

A certificate of location of a mining claim describing it as situated about two miles from a certain town, was held not to constitute a substantial compliance with the statute requiring a description of the location of the claim with reference to some natural object or permanent monument.

Ford *v.* Campbell, (Nev.) 92 Pac. 206. (November, 1907.)

LOCATION OF MINING CLAIM — SUFFICIENCY OF CERTIFICATE.

The words "necessary to hold claims" in the statute providing that the certificate of location and of labor and improvement need not be sworn to, did not refer to "certificates of location," but only to the words "labor and improvements," which referred to the statute requiring the expenditure of one hundred dollars annually in labor and improvements in order to hold a mining claim prior to the issuance of patent.

Ford *v.* Campbell, (Nev.) 92 Pac. 206. (November, 1907.)

MINING CLAIM — ABANDONMENT — RELOCATION.

A mining claim was located and the locator performed certain work which on survey was discovered to be on conflicting territory, whereupon the locator posted a relocation notice to the effect that such work had been abandoned. The first notice was not removed from the location monument, and the notice of relocation stated that the claim was relocated in order to better describe the place of the lode claim. These facts were held insufficient to indicate an abandonment of the rights under the prior location.

Ford *v.* Campbell, (Nev.) 92 Pac. 206. (November, 1907.)

CONFLICTING RIGHTS.

MINING CLAIM — WIDE LODE — APEXING VEINS.

The senior location was held to take the entire width of the vein on its dip, where the apex of such vein was partly within two or more adjacent lode mining claims.

Lawson *v.* United States Min. Co., 28 Superior Court Reporter 15. (October, 1907.)

LOCATION OF MINING CLAIM — EFFECT OF PATENT.

The issue of patents for lode mining location was a recognition by the Land Department of the conformity of the proceedings to the local rules and customs of the district, and such ruling could not be challenged by third parties claiming rights which arose subsequent to the notices.

Lawson *v.* United States Min. Co., 28 Superior Court Reporter 15. (October, 1907.)

LOCATION — PRIORITY — CONCLUSIVENESS OF PATENT ON WIDTH OF VEIN AND APEX.

Priority of entry and patent alone does not conclusively establish seniority of location, so as to give the holder of a lode mining claim under such a patent the right to the entire width of the vein on its dip, where part of the apex of such vein is within such claim and part of it within an adjoining claim.

Lawson *v.* United States Min. Co., 28 Superior Court Reporter 15. (October, 1907.)

ADVERSE SUIT — PRESUMPTION AS TO SUBTERRANEAN RIGHTS.

There is no presumption that subterranean rights under lode mining locations were considered and determined in an adverse suit in the absence of the record of such suit.

Lawson *v.* United States Min. Co., 28 Superior Court Reporter 15. (October, 1907.)

ADVERSE ACTIONS — TIME OF COMMENCING.

Under the statute an adverse action must be commenced within thirty days after the filing of the adverse claim; and such time is determined by the statute providing that a civil action shall be commenced by the filing of the complaint and the issuance of a summons, etc.

Harris *v.* Helena Gold Min. Co., (Nev.) 92 Pac. 1. (October, 1907.)

MINING LEASES.

CONSTRUCTION OF MINING LEASE — ANNUAL ROYALTY — FORFEITURE.

An agreement in a lease by which the lessee agreed to pay to the lessor, a specified sum per annum for ore mined during the life of the lease, as a minimum royalty for the demised property, without reference to the quantity of ore actually mined, and providing that all rents and royalties agreed to be paid could be deemed and treated as rents reserved upon contract by the lessor, was held an absolute undertaking to pay the stipulated annual sum. And a clause providing that on failure of the lessee to make the payment of the royalties or to develop the property, it should become forfeited and utterly void, did not sufficiently manifest an intent that it should be optional with the lessee to develop the property and to pay the rent or not, as he might see fit, as the clause of forfeiture was intended for the benefit of the lessor.

Lawson *v.* Williamson Coal & Coke Co., (W. Va.) 57 S. E. 258. (April, 1907.)

OIL AND GAS LEASE — PRIORITY — NOTICE.

A lessee accepting a subsequent oil and gas lease with notice of a prior lease and with knowledge of the fact that valuable improvements

had been made under such former lease, and that the lessor had accepted its benefits and consideration, holds subject to such prior lease and will not be permitted to question the validity of such prior lease on the ground of want of mutuality, or that it had been revoked by the giving of the subsequent lease.

Compton *v.* People's Gas Co., (Kans.) 89 Pac. 1039. (April, 1907.)

PAROL GRANT OF MINING PRIVILEGE — LICENSE — REVOCATION.

A parol grant of mining privilege on lands is a mere license, and is revocable at will, and conveys no title to any ores not severed and reduced to possession before such revocation. The expenditures of moneys in mining under a parol license adds nothing to the grant.

McCullough *v.* Rains, (Kans.) 89 Pac. 1041. (April, 1907.)

LEASE OF COAL — PLACE OF PAYMENT OF ROYALTY.

Where no place was specified for the payment of royalties in a coal lease, they must be demanded on the premises before there can be any forfeiture for the nonpayment.

Drake *v.* Pennsylvania Coal Co., (Pa.) 66 Atl. 660. (April, 1907.)

COAL LEASE — OPTION — FORFEITURE.

A lease to mine coal and other minerals and pay royalty thereon to the lessor, provided that the mining was to be done at the lessee's discretion, and that no cessation of operations or failure of the lessee to avail himself of the privileges conferred by the lease should operate as a forfeiture thereof. This was held to be a mere option, based upon no consideration, and could be withdrawn at any time before money was expended in doing what was optional upon the part of the lessee.

Collins *v.* Smith (Ala.), 43 So. 838. (April, 1907.)

DEED FOR GAS AND OIL — CONSTRUCTION — TITLE.

A deed granting all the gas and oil under certain described lands, together with the exclusive right to enter thereon for the purpose of drilling and operating for gas and oil, but which limited the estate granted to a term of seven years, and as much longer as gas and oil might be found in paying quantities, and which further stipulated for the payment of commutation money for delay in drilling, and providing for the delivery into pipe lines for the grantor of one eighth of all the oil produced, and for the annual payment of rental for every gas well from which gas was used, was held not to pass title to the gas and oil in place, but the title thereto remained in the grantor.

Toothman *v.* Courtney (W. Va.), 57 S. E. 415.

CONSTRUCTION OF LEASE — PAYMENT OF ROYALTY.

Where unforeseen difficulties in the operation of a mine prevented the lessee from operating it at a profit or in competition with other mines in the particular vicinity, the lessee was held not liable for the payment of a specified royalty for coal not actually mined, where the lease provided

that payment should be made, unless the lessee was unavoidably prevented from taking out the coal.

Wilson *v.* Big Joe Block Coal Co. (Iowa) 112 N. W. 89. (June, 1907.)

LEASE OF MINE — FAILURE TO WORK MINE — FORFEITURE.

A mining lease provided that the lessee should have one year to explore for ore, and immediately thereafter he should commence mining operations, and if he failed to prosecute the operations, and during the remainder of the term failed to mine, for a continued period of one year, and raise the ore on which royalty was payable, the lease should be void at the option of the lessor except as to the liability of the lessee therein. It is further provided that after one year from the date of the lease that the lessee must mine at least one thousand tons of coal annually, or pay the royalty on that amount. An action to forfeit the lease could be maintained after one year on the lessee's failing at any time to prosecute the mining operations for a year; and the acceptance by the lessor of the minimum royalty was not a waiver of the right to forfeit the lease for the failure to mine for such year.

Chauvenet *v.* Person (Pa.), 66 Atl. 855. (April, 1907.)

LEASE OF MINE — CONDITION.

The lease of a mine provided that the lessee should leave sufficient pillars to support the roof, and for any breach of this condition the lease should be forfeited. The court decided that this stipulation was not a mere covenant, but a condition, the violation of which defeated the estate of the lessee, at the option of the lessor.

Greer *v.* Boston, etc., Zinc Co., (Mo.) 103 S. W. 151. (June, 1907.)

LEASE OF MINE — NOTICE — FORFEITURE.

A lease of mine provided that in certain event the lease should be forfeited at the option of the lessor and that notice of his option should be given by mail. In an action to forfeit the lease, it was held that notices sent by mail to the lessee at his place of business, followed immediately by the peaceable entry by the lessor was sufficient to terminate the tenancy.

Greer *v.* Boston, etc., Zinc Co., (Mo.) 103 S. W. 151. (June, 1907.)

LEASE OF MINERAL LANDS — FAILURE TO WORK — FORFEITURE.

A lease or conveyance of coal and other minerals with the right to mine the same for ninety-nine years and a renewal in perpetuity, and by the terms of which the grantee was to render an account and pay royalties at specified rates when any coal or other mineral was mined, was held to contemplate a mining enterprise, and imposed on the grantee the obligation to make reasonable effort to operate such mines and a failure to do so for more than twenty years amounted to a forfeiture by abandonment.

Brown *v.* Wilmore Coal Co., 153 Fed. 143. (April, 1907.)

MINING LEASE — EXECUTED BY LESSOR — CANCELLATION

A coal and oil lease, running for a long period of time, provided for a specified royalty, and that the operation of the mines should be begun and continued at the discretion of the lessee and that no cessation of operations in mining should work a forfeiture; the lease was signed by the lessor only. In an action to forfeit the lease for failure to operate the mines, it was held that the lease was a mere unilateral contract, subject to cancellation at the instance of the lessor, and that the lessee having taken no steps to operate the mines thereunder for more than three years could claim no rights by virtue of the lease.

Collins *v.* Abel, (Ala.) 44 Southern 109. (May, 1907.)

LANDLORD NOT LIABLE FOR CAVING THROUGH WILFUL ACTS OF TENANTS.

As between two lessees of different portions of a mine each must know that the only use to which the ground can be put is mining; and the landlord is not liable for the wilful or negligent acts of the tenant of the upper level and was not required to take active measures to prevent the caving of the ground of the upper level to the injury of the tenant and lessee of the lower level. In such case, it is the duty of the tenant or lessee to require from the mine owner such reasonable stipulations as will protect him. In order to hold the mine owner liable in such case, it must appear that he made the lease to the lessee of the upper level after leasing the lower, and that the actual situation was then such that no other result could have followed from working the upper ground but material injury to the lower, and that such state of affairs was known to the mine owner when the upper ground was leased.

Peterson *v.* Bullion-Beck, etc., Min. Co., (Utah) 91 Pac. 1095. (September, 1907.)

MINING — UPPER AND LOWER LEVELS.

The law does not presume, merely because one tract of ground is above the other, that the working of the upper will injure the lower; the lessee of the lower level is bound to protect his own rights in making a contract against the operations of one above him, if the situation is such as to require it, rather than to rely upon a supposed duty of the lessor or mine owner, to protect him at all hazards.

Peterson *v.* Bullion-Beck, etc., Min. Co., (Utah) 91 Pac. 1095. (September, 1907.)

DEED — GAS AND OIL LEASE.

A deed granting all the oil and gas under a certain tract of land, with the exclusive right to enter at all times for the purpose of drilling and operating for oil and gas, but limiting the estate to a term of years and as much longer as oil or gas should be found in paying quantities, and stipulating for penalty in case of delay in drilling, and further providing for the delivery into pipe lines for a certain share for the guarantor of all the oil produced, and for the payment of a yearly rental for each gas well, was held not to pass the title to oil and gas in place; and was in

legal effect only a mining lease and that the title to the oil and gas in place remained in the grantor.

Tootman *v.* Courtney, (W. Va.) 58 Southeastern 915. (October, 1907.)

ASSESSMENT WORK.

MINES — ASSESSMENT WORK — EXPENSE OF KEEPER.

In an action to quiet title to certain mines where the defendant claimed that the plaintiff had abandoned such mines, it was held that a keeper was necessary and that his expenses were properly a part of the assessment work. But it was said that where a keeper was maintained simply to comply with the law relative to assessment work and to hold the property without any intent within a reasonable time to make use for the purpose of mining such structures as there may be thereon, and which he was employed to care for, then such expenditure should not be counted as assessment work. The expenditure, to count as assessment work, must be made in good faith, and it must reasonably be of personal use and benefit to the property as a mine by guarding valuable improvements made therein or thereon against deterioration or destruction, when such improvements may reasonably be said to be of value to the property as a mine.

Kinsley *v.* New Vulture Min. Co., (Ariz.) 90 Pac. 438. (March, 1907.)

LIABILITY OF INJURY TO MINER.

OPERATING MINE WITHOUT LICENSED FOREMAN — EFFECT ON LIABILITY — TENNESSEE.

The provisions of the Tennessee statute permitting a mine to be operated for thirty days without a licensed foreman, without being subject to penalty, was held not to give an unlicensed foreman the standing of a lawful foreman under such a statute, during the first thirty days of his employment, even though he was subsequently licensed; but that until so licensed the owner or operator was in control of the operation of the mine as before the enactment of the statute, and his liability was not affected thereby.

Cumberland Coal, etc., Co. *v.* Gray, 152 Fed. 939. (March, 1907.)

MASTER AND SERVANT — DELEGATED DUTIES — FELLOW SERVANTS.

The pit boss of a mine may delegate to a miner the duty of making an inspection. And where such duty was delegated to a miner, while performing such duty, he was as to his co-workers a vice principal and not a fellow servant, irrespective of his ordinary duty. And a miner may recover for injuries which resulted from the negligent performance of duties of the person to whom the power of the pit boss was so delegated.

Western Coal, etc., Min. Co. *v.* Buchanan, (Ark.) 102 S. W. 694. (April, 1907.)

MINING PLANT — DANGER SIGNALS REQUIRED — ILLINOIS.

A hoisting engine used at the top of an incline for the purpose of drawing rock and coal cars to haul material from the mine, though outside of the mine, was a part of such mine, under the statute of Illinois which made it the duty of the mine manager to see that all dangerous places above and below were properly marked, and danger signals displayed when required, and the mine owner was liable for injury to a miner for failure to have such hoisting engine inspected and marked dangerous as required by the statute.

Spring Val. Coal Co. *v.* Greig, (Ill.) 80 N. E. 1042. (April, 1907.)

RELATIVE DUTY OF MASTER AND SERVANT.

The general rules as to the relative duty of master and servant as to knowledge of danger applies to miners and mine owners. But in all such cases they do not stand upon an equal footing, even when they have equal knowledge of the danger. The position of the servant is one of subordination and obedience to the master, and he has the right to rely upon the superior skill of the master, and he is not entirely free to act upon his own suspicions of danger. Where a servant, being ordered to work in a particular place, obeys and is injured, he will not be held to be guilty of contributory negligence, unless the danger is so glaring that a reasonably prudent person would not have entered into it.

Pioneer Min., etc., Co. *v.* Smith, (Ala.) 43 So. 561. (April, 1907.)

EMPLOYER'S LIABILITY FOR INSUFFICIENT REPAIRS.

A defect in the roof of a mine entry was reported to the mine owner, who thereupon attempted to remedy it, and then assured the miners that the roof was no longer dangerous. A miner while at work in the mine was afterwards killed by the falling of that part of the roof where the repairs had been made. It was held that the failure of such miner to examine the roof after the repair was not negligence as a matter of law, where it was not shown that he knew of the danger, or that the defect was open and glaring to ordinary observation.

Pioneer Min., etc., Co. *v.* Smith, (Ala.) 43 So. 561. (April, 1907.)

DUTY TO FURNISH PROPS.

Under the Illinois statute the operator of a coal mine is required to provide miners with a sufficient supply of props, caps and timber on demand, and this duty is not discharged by the operator furnishing such props, etc., as he himself deems sufficient, but he must furnish any particular kind on a reasonable and timely demand by a miner.

Springfield Coal Min. Co. *v.* Gedutis, (Ill.) 81 N. E. 9. (April, 1907.)

NEGLIGENCE — FAILURE TO INSTRUCT MINER.

Where the drilling out of unexploded charges was shown to be highly dangerous, and in the proper conduct of mining should never be done, it was held to be negligence of the company for its superintendent to direct an inexperienced miner, without instruction, to drill out an unexploded charge by reason of which he was injured.

Peters *v.* George, 154 Fed. 634. (May, 1907.)

DEATH OF MINER — NEGLIGENCE.

In an action for the death of a miner while descending into a mine, by reason of the breaking of the cable attached to the bucket in which he was being lowered, the question of whether the operator of the mine was negligent in furnishing a defective cable was for the jury to determine, and the determination of the jury that he was so negligent, would not be disturbed by the court of appeal.

Owen *v.* Retsof Min. Co., 104 N. Y. S. 37. (May, 1907.)

INJURY TO MINER — ASSUMPTION OF RISK.

A miner who went from his usual place of work in the mine to another part of the mine to get some tools he had formerly loaned to another employe, assumed the risk incident to the place to which he went, and was not entitled to recover for injuries occasioned by the falling of slate from the roof of that part of the mine.

Pioneer Min. & Mfg. Co. *v.* Talley, (Ala.) 43 So. 800. (May, 1907.)

INJURY TO MINER — PROXIMATE CAUSE.

A miner, while attempting to remove a car which had been stopped by reason of a defect in the track, was run into and injured by another car which the foreman of the mine was pushing along the same track. It was held that the defect in the track was the proximate cause of the injury and not the act of the foreman, and that the mine owner could not escape liability on the ground that the accident was so remote a consequence of the defect as not to be expected with any degree of reason.

Moore *v.* Royal Lead & Zinc Co., (Kans. App.) 102 S. W. 616. (May, 1907.)

INJURY TO MINER — EXPERT EVIDENCE.

Where a miner was injured by a fall of rock where he was working, in an action for damages it was held proper for him to prove by expert evidence the usual methods followed in the protection of the miners from danger by falling rock.

Spencer *v.* Bruner, (Mo.) 103 S. W. 578. (June, 1907.)

INJURY TO SERVANT — FELLOW-SERVANT — VICE-PRINCIPALS.

The owner of a coal mine was held liable for injuries to a servant caused by an explosion of dynamite in the roof of the mine, placed and ignited by timbermen, whose duty it was to keep the mine roof in safe condition. And it was competent to prove the customary hours for firing shots by the timbermen in the entries of the mine, and that they were not usually shot at the time when the miner was injured, thereby showing that the injured miner exercised due care and that he had no reason to expect danger from such a source.

Donk Bros. Coal, etc., Co. *v.* Thil, (Ill.) 81 N. W. 857. (June, 1907.)

MASTER AND SERVANT — RISK ASSUMED.

A pumpman engaged at the bottom of a shaft in a mine, who knew that the shift boss performed the duties of inspector, thereby assumed the risk of the inspection system under which the mine was worked.

Williams *v.* Verona Min. Co., (Mich.) 112 N. W. 496. (July, 1907.)

FELLOW SERVANTS — LIABILITY FOR NEGLIGENCE.

A pumpman engaged at the bottom of a shaft in a mine was held to be a fellow-servant of the brakeman in charge of the hoisting cage, and that the mine owner was not liable for the death of such pumpman by reason of the negligence or wrongful act of such brakeman.

Williams *v.* Verona Min. Co. (Mich.) 112 N. W. 496. (July, 1907.)

INJURY TO MINER — EVIDENCE AS TO CONDITION OF MINE.

Where a miner was injured by a fall of rock from the roof of a mine in which he was working, it is proper to show, in an action for damages, that the mine was inherently dangerous, and that skillful operators should have known of such condition and taken precaution against it. But it must appear that the conditions at the time of the accident were the same as those at the time to which the testimony related.

Arris *v.* Standard Plaster Co., 105 N. Y. Sup. 440. (July, 1907.)

MINE OWNER — DUTY TO SUPPORT ROOF OF MINE — MINE BOSS.

It is the duty of the owner or operator of mines to use proper care in placing and setting timbers to support the roof of the mine and keep them in proper condition. This duty extends also to keeping the roof over the entries and to the roof over the necks of the rooms safe and secure as well as the roof of the mine itself. And for this purpose it is the further duty of such mine owner or operator to provide a competent mining boss to inspect the walls and roof of the mine and such supports, and to see that they are not only properly set but to exercise a continued inspection and that they are kept sufficient to render the premises safe.

Brunson *v.* Southwestern Development Co., (Ind. Terr.) 104 Southwestern 593. (September, 1907.)

DEATH OF MINER — MINE OWNER NOT NEGLIGENT.

Where a miner had himself worked out the room and after part of the roof had fallen, had himself set a prop at such place in a workman-like manner, sufficient to render the roof reasonably safe, such miner had the option to do his own timbering for pay, or to have the company's timberman do the work. It appeared also that the foreman had regularly inspected the room, and had found it free from signs of weakness on the day before the accident. Under such circumstances, it was held that the company was not negligent and not liable for the death of the miner caused by falling of the roof.

Brunson *v.* Southwestern Development Co., (Ind. Terr.) 104 Southwestern 593. (September, 1907.)

INJURY TO SERVANT — ASSUMPTION OF RISK.

A mule, driven by a miner in hauling cars of coal out of a mine, took fright at an object properly in the mine, broke loose from the car and ran away. The pit boss represented to the driver that the mule was only frightened; that ordinarily he was gentle and could again be worked and induced the driver to take it back into the mine, hitch it up and finish the work; the miner did so, under protest, and relying upon the opinion of the pit boss as to the gentleness of the mule, and the safety of the undertaking. He continued driving the mule, hauling out a number of loaded cars, when later in the day the animal again became frightened at the same object, by reason of which the driver was thrown under the car and received serious injury. It was held that the miner assumed the risk with knowledge of the danger and could not recover.

Milby, etc., Min. Co. *v.* Ball (Ind. Terr.) 104 Southwestern 76. (September, 1907.)

STATUTORY DUTIES OF MINE OWNERS — ASSUMPTION OF RISK.

Where a mine owner failed to comply with the statute, requiring him to guard his employee from injuries by reason of dust, bad air, explosives, etc., he was not permitted to plead contributory negligence or assumption of risk in an action by a miner for an injury caused by failure to comply with the statute.

Bolen-Darnall Coal Co. *v.* Williams, (Ind. Terr.) 104 Southwestern 867. (September, 1907.)

INJURY TO MINER — MEASURE OF RECOVERY.

Where a miner was burned in an explosion because the owner failed to comply with a statutory requirement, he was entitled to recover for permanent disability and inability to earn a living, for his pain and suffering, and also for the humiliation of being handicapped through life by distorted features and body.

Bolen-Darnall Coal Co. *v.* Williams (Ind. Terr.) 104 Southwestern 867. (September, 1907.)

MINER — NOTICE OF DEFECTS.

While a miner, like other servants was bound to take notice of manifest defects in the entry of the mine where he was working, the law does not require him to make an examination for hidden defects, but he may assume that the owner or operator had used reasonable care not to endanger his position by other workings.

Superior, etc., Min. Co. *v.* Kaiser (Ill.) 82 Northeastern 239. (October, 1907.)

MINER — ASSUMPTION OF RISK.

The rule that a servant or a miner assumes the ordinary risks incident to the work in which he is engaged, is based on the supposition that the master has performed the duties of care, caution, and vigilance which the law places on him.

Superior, etc., Min. Co. *v.* Kaiser, (Ill.) 88 Northeastern 239. (October, 1907.)

NEGLIGENCE — FAILURE TO INSPECT.

The failure of the owner and manager of a mine to thoroughly examine and inspect the face of a certain cross-cut was held to constitute such negligence as to permit a miner to recover for injuries received while working in an entry near to such cross-cut, and that such cross-cut was developed in advance of and so close to the rib and face of the entry as to loosen, crack, and weaken the rib and face of such entry at the point where the miner was at work and where he was injured. Such miner was not charged with contributory negligence where the fall of coal which injured him was from the face of his entry, which, on examination, appeared to be safe on the morning of the accident.

Superior, etc., Min. Co. *v.* Kaiser, (Ill.) 82 Northeastern 239. (October, 1907.)

FIRING BLAST — VOLUNTEER.

Where blasting was required to be done after the general work in the mine ceased, and where it was the custom, when one miner left his work before quitting time, for his assistant to fire off his blast, it was held that such an assistant in firing the blast of his fellow miner was not a volunteer, and that in case of injury he was entitled to recover.

McHenry Coal Co. *v.* Render, (Ky.) 104 Southwestern 996. (November, 1907.)

NEGLIGENCE — ACCUMULATION OF GAS.

Proof that a mine was not properly ventilated, that the air was bad, and that an explosion of gas occurred, was held sufficient proof of negligence on the part of the mine operator.

McHenry Coal Co. *v.* Render, (Ky.) 104 Southwestern 996. (November, 1907.)

EXPLOSION OF GAS — PERSONAL INJURIES — DAMAGE NOT EXCESSIVE.

Where it appeared that a miner by reason of an explosion of gas was thrown some distance down the entry and bruised and was severely burned, a verdict of six thousand dollars was held not excessive and such verdict could not be set aside on appeal because the evidence of the defendant indicated that there was no explosion of gas but that the plaintiff was burned by a blast which he exploded.

McHenry Coal Co. *v.* Render, (Ky.) 104 Southwestern 996. (November, 1907.)

MISCELLANEOUS.

MINING — PARTNERSHIP.

Where different owners of a mine united in working it without any agreement, the act of working together created a partnership.

Kirchner *v.* Smith, (W. Va.) 58 So. 614. (February, 1907.)

MONEY ADVANCED TO PURCHASE MINE — SPECIFIC PERFORMANCE.

A person advanced money to purchase certain mines under an agreement that he should receive a proportionate share of the capital stock of

a corporation to be formed for the purpose of operating such mines. On a failure to issue and deliver the proportionate amount of the capital stock after the corporation was organized, it was held that the person so advancing the money might have specific performance of the agreement, where it appeared that the stock had no known value and that the damages were incapable of ascertainment.

Rau *v.* Seidenberg, 104 N. Y. S. 798. (March, 1907.)

AGENT — RIGHT TO PURCHASE MINING PROPERTY AT TAX SALE.

An agent appointed to care for mining property and its title for a corporation of which he was a stockholder, and to accept service of process, but was given no funds to pay taxes or other expenses, and was without authority to advance money for the corporation, after repeated notices by him to such owner that taxes were due, that the property had been advertised for sale at tax sale, was held to have the right to bid in a part of the property after most of it had been sold for taxes, and thereafter obtained a tax deed and thereupon notified the corporation, offered to convey if it would reimburse him, and subsequently sold the property which was then wholely unproductive. The original owner was held not entitled to recover the property six years later after valuable ore had been discovered.

Steinbeck *v.* Bon Homme Min. Co., 152 Fed. 333. (March, 1907.)

SUPPORT OF SURFACE.

The owner of the mineral estate may by contract with the owner of the surface relieve himself from the duty of supporting the surface, and from all liability for any injury done to it by mining and removing the minerals from the mine below.

Miles *v.* Pennsylvania Coal Co., (Pa.), 66 Atl. 764. (April, 1907.)

MINERS' LIENS — LEASE — MINING RECORDS.

A statute authorized liens on mines in favor of laborers and material, but provided that it should not apply to the owner when the mine was worked by a lessee, if a copy of the lease was recorded in the mining records of the county before the work under the lease was begun. In the absence of a statutory definition of "mining record" or of any designated record in which such instrument should be recorded, it was held sufficient to record such lease in a book kept by the county clerk for the recording of leases and instruments effecting the title to mining claims, and which was designated "mining conveyances."

Slover *v.* Bailey (Ore.), 90 Pac. 665. (June, 1907.)

OWNERSHIP OF COAL IN LANDS CONDEMNED FOR PUBLIC USE.

Under a statute providing for the appropriation of lands for public use without the consent of the owner, and providing that the title should remain in the owner subject to the use for which it was taken, the owner of land taken for a right of way of a railroad, had the right to remove coal from under the right of way and to connect such coal by tunnels, but in so doing he must not impair the surface for the use of such right of way by removing the support.

Eldorado, etc., R. Co. *v.* Sims (Ill.), 81 N. E. 782. (June, 1907.)

SHORTAGE OF RAILROAD CARS — ALLOTMENT TO MINES.

In case of unavoidable shortage of cars, a railroad company may distribute such cars as it has to the several mines on its line of road on a percentage basis, calculated on the production of the several mines. The mine owners are only entitled to their proper percentage of available cars; but such distribution cannot interfere with the right of individual owners to the exclusive use of their own cars. And the railroad company may allot an arbitrary number of cars for development to new mines which have no basis for a percentage.

United States *v.* Baltimore, etc., R. Co., 154 Fed. 108. (June, 1907.)

STOCK ISSUED IN PAYMENT OF CLAIM.

A mining corporation may issue stock to an attorney in payment for services rendered, and in the absence of fraud, it cannot be questioned by a stockholder.

Bogeler *v.* Punch, (Mo.) 103 S. W. 1001. (June, 1907.)

RELIEF BY STOCKHOLDERS — EFFECT OF DELAY.

Where stockholders of a mining corporation applied to a court of equity for protection of their rights against the alleged wrongful acts of the directors, they could not recover because they had failed to act with reasonable diligence, or present some excuse for not having done so.

Jones *v.* Bonanza Min. etc., Co., (Utah) 91 Pac. 273. (July, 1907.)

RULE AGAINST STRIKING MINERS ENTERING MINE — REASONABLENESS.

A mine owner may adopt a rule prohibiting persons who have left his employ from entering his mine; and such rule is reasonable although adopted for the purpose of keeping striking miners out of the mine, and though it prohibits them from entering the mine for their tools.

Sloss, etc., Iron Co. *v.* Prior (Ala.) 44 So. 649. (July, 1907.)

FIXTURES — MACHINERY.

Mining houses and machinery erected with the intention that they should remain and become a part of the real estate are fixtures.

Hoye Coal Co. *v.* Colvin (Ark.) 104 S. W. 207. (July, 1907.)

METALLURGICAL ABSTRACTS.

By WILLIAM CAMPBELL.

Antimony.

Antimony Smelting, Notes in: G. Pautrat. *E. and M. J., 84, 493.* Smelter at Mayenne, France. (1) Rich ore 50–60 per cent. Sb, (2) poor 10–20 per cent. Sb. Rich used for metal, poor for oxides. Operation — fusion with scrap iron: refining fusion. Both in reverberatory furnaces.

Bismuth.

Electrolytic Refining of Bismuth: A. Mohn. *Electrochem. and Met. Ind.*, V., 314. Describes some work on parting of a Mexican lead-bismuth bullion, 14.5 per cent. Bi, 1.58 per cent. Sb, 1.46 per cent. Fe, AuAg 0.4 per cent. First, the lead was refined by Betts process. Second the slimes were purified. Third the bismuth refined electrolytically and fourth electrolytic refining of gold and silver.

Copper.

The McDougall Roasting Furnace: L. S. Austin. *M. and S. P.*, XCV., 280. Description and working: three illustrations.

The Charging of Blast Furnaces: E. H. Messiter. *M. and S. P.*, XCV., 528. Describes the ore bedding system (Messiter) manufactured by Robbins Belt Co. It consists of receiving hoppers, crushing and sampling, a bedding and storage plant, devices for delivering the charge from the belt conveyor system to the furnaces. The subject is taken up in detail.

Negative Results in Pyritic Smelting: G. F. Beardsley. *E. and M. J.*, 84, 343. Account of attempts made in 1906 to treat copper nickel ores of Sudbury by pyritic smelting. The pentlandite is supposed to have liquated and carried with it a certain amount of pyrrhotite for a low matte (12–15 per cent.) was obtained after furnace had been run a short time. CuNi mattes are very cutting, especially to iron and steel.

Lake Copper versus Electrolytic: A. M. Fairlie. *Metal Ind.*, V., 1, 43, 71, 103, 138. Gives monthly prices for 1905–06. Electrolytic copper slightly cheaper than lake. At Buffalo the Calumet and Hecla Mining Co. have an electrolytic plant (Buffalo Smelting Works) to recover silver and remove arsenic from certain lake copper. This is the only plant for preparing electrolytic copper from native. A map of copper centers of United States and Mexico is given. The paper deals with the metallurgy of copper in an elementary manner.

The Washoe Smelter: M. and M., 28, 37. Abstract from Bull. 33 of Anaconda Copper Mining Co. Sample mill, roaster building, reverberatory and blast furnace buildings described. Refining and casting. Power. Arsenic plant.

Washoe Reduction Works: M. and M., 28, 131, 248. Gives flow sheet of plant, sections of McDougal Roaster and Smelting Reverberatory and descriptions of their working. Briquetting, blast furnaces, converters, etc. Photographs of different sections.

Copper Smelting in Utah: R. B. Brinsmade. *M. and M.*, 28, 178. Describes practice at Utah Con., Murray, U. S. and Bingham Con. at Bingham Junction and Yampa in Lower Bingham. Illustrated by several photographs.

Notes on the Fundicion Smelter: M. and S. P., XCV., 91. Douglas Copper Co. erecting a smelter near Guaymas. Will be 600 tons capacity. Power and Mining Mach. Co.'s furnaces 44 in. × 160 in. × 14 ft. column. 15 ft. by 5 ft. settlers. Vezin sampler cuts one tenth of total tonnage for sampling works.

Mining and Metallurgy in South Wales: E. Walker. *E. and M. J.*, 84, 57. Coalfield. Copper smelting in the Swansea district: Vivian & Sons at Hafod, Williams, Foster & Co., at Morfa Works, Swansea, Elliott's Metal Co. at Pembrey, Cape Copper Co. at Briton Ferry, and Rio Tinto Co. at Port Talbot. Lead smelters: Vivians' White Rock Works, Landore and Elliott's Metal Co. at Pembrey. Nickel plants: Anglo-French Co. at Swansea, Mond Nickel Co. at Clydach.

Mansfield: Operations and Tendencies of Modern: Wagner & Primrose. *E. and M. J.*, 84, 671. Hand picking. Roast yard at Krughütte 40 m. × 5–7 m. wide, 2 m. high. Smelt to 40 per cent. matte in circular pilz, 5 lb. pressure. Some metallic copper formed, containing Au, Ag, etc. 40 per cent. matte sent to roasting furnaces in the Spurhutte, hexagonal kilns, smelt to 75 per cent. matte in reverberatories. Three methods of further treatment: (1) Ziervogel, (2) New Welsh process to blister and electrolytic refining, (3) New Gunther process. Electrolysis of 75 per cent. matte directly.

Briton Ferry Works of the Cape Copper Co.: E. Walker. *E. and M. J.*, 84, 304. Owns mines in Namaqualand, Cape Colony: Ore from Tilt Cove, N. F. Total output in 1906 was 10,812 tons, of which 793 tons were sulphate. Ores and matte calcined. Smelted in reverberatory to 76 per cent. white metal. White metal refined by Nicholl's direct process. Calciners are 60 soft cylinders and traveling rake type (Prosser type) gas-fired. Tilt Cove ores, 3½ per cent. Cu, 35 per cent. S, are burnt in sulphuric acid kilns. Burnt ore to 4 cupola furnaces with reverberatory slags. The matte produced is run in a reverberatory for copper bottoms. Bottoms granulated and used for copper sulphate manufacture. Au as slimes.

Refinery of Rio Tinto Co., Port Talbot, Wales: E. Walker. *E. and M. J.*, 84, 111. Bessemerizing now done in Spain, refining in Wales. Moved from old site at Cwmavon to Port Talbot docks. Two Temperley transporters. Five refining furnaces, each 30 tons; 20,000 tons refined per an. Refinery slags run 4–5 per cent. Cu and are smelted in a small water-jacket blast furnace. Slag and copper cast direct into moulds on trucks.

How to Outline an Electrolytic Copper Refining Plant. C. C. Christensen. *E. and M. J.*, 84, 127. Gives some figures on the subject.

GOLD.

Present Status of the Gold Mining Industry: J. H. Curle. *M. and S. P.*, XCV., 147. Transvaal ore yields about $8 a ton, working costs $6; 8,000 stamps. Future of the field. Rhodesia only good for small syndicates. West African gold mines; conglomerate bed of Wassau. Nevada. Canada (Klondike). Mexico, El Oro. Australia. India. Myzore crushes 15,000 tons a month, yielding over $20 a ton. Adjacent Champion Reef same to below 2,000 ft.

Mining in Australia: W. J. Loring. *M. and S. P.*, XCV., 501. General article.

Gold Mining in Egypt: C. S. Herzig. *M. and S. P.*, XCV., 212. Ancient method of grinding. Tails show 88 per cent. passed 200-mesh. Most of gold extracted. Present mines not paying. Photographs show ancient grinding stones.

The Nomenclature of Modern Placer Mining: J. P. Hutchins. *E. and M. J.*, 84, 293. Alluvial placers, diluvial placers, those formed in situ. Methods of exploitation are taken up. See

Essential Data of Placer Investigation, E. and M. J., 84, 339, 385.

Notes on Hydraulic Mining: Mines and Minerals, 28, 1. Only one hydraulic mine in California working to-day, the La Grange works on 4½ c. per cu. yd. The Cariboo district, B. C., is being opened up. Gravels run 10 c. cu. yd. to be worked. Average value is 20 c. and cost 8 c. Several illustrations of placers being worked hydraulically.

Hydraulic Mining in Cariboo: D. Waterman. *M. and S. P.*, XCV., 302. Illustrated article on the Consolidated Cariboo Hydraulic Mining Co. Ltd. at Bullion, on South fork of Quesnelle River, B. C.

Hydraulic Equipment of the Old Channel Mines · J. M. Nicol. *M. and S. P.*, XCV., 335. Work at 3 c. per cu. yard; 7,000 cu. yards per 24 hours.

Experiences with Gold Dredges at Ruby, Mon: J. P. Hutchins. *E. and M. J.*, 83, 1223, 84, 69. Description of dredges; 7.5 cu. ft. buckets weighing 1,400 lbs.; 12.5 cu. ft. buckets weighing 2,800 lbs. Bucket lips of manganese steel. Dredge ladders. Trommels, sluice with quicksilver, 66 in. wide and 135 ft. long. Clean up every 10 days.

Old Methods in Mexico: M. and S. P., XCV., 372. Extracts from a report of E. Tillman, Royal Prussian Commissioner of Mines in 1865, on conditions in Guanajuato. Gives table of output of Mexican mints from 1690 to the declaration of independence in 1822.

The Desert Mill: A. R. Parsons. *M. and S. P.*, XCV., 494. 100 stamps, stem 427 lb. tappet 140, boss 320, shoe 180 lbs. Drop 1–3–5–2–4. Height 6½ in., 104 a minute, duty 4.34 tons through 12-mesh. Crush in 0.15 per cent. KCN solution. Pulp to ten 20-in. double cone classifiers — 10 No. 5 Wilfley concentrators. Coarse to five 5-ft. Huntington mills, 30-mesh, 10 Wilfley concentrators. Concentrates to smelters. Tails to classifiers, divided into slime and sand. Blaisdell equipment and Butters filters described. Four excellent illustrations. Flow sheets.

Mining Conditions in S. Africa: J. B. Pitchford. *M. and M.*, 28, 49. Deals with mining. 7,000 stamps on the Rand. 320 at Simmer and Jack and 400 at Summer East and Knights Deep Joint mill: Robinson gold mine has 220 and 3 Allis-Chalmers tube mills. Concentration. Frue vanners or Wilfley tables. Cyanide plant.

Cyanidation in the Transvaal: M. and S. P., XCV., 78. Correspondence in defence of Denny Bros' "new metallurgy" of all-sliming. *M. and S. P.*, XCV., 111. A letter from G. A. Denny on the process.

Working Costs on the Rand and Comparisons with Mines in California: R. E. Browne. *M. and S. P.*, XCV., 113. Abs. from S. A. Ass. Engs., June 1, 1907. Gives tables comparing costs of Oneida mine on the Mother Lode, the Empire mine at Grass Valley and the Rand.

Cyanide Practice at the Homestake Mills: Bosqui. *M. and S. Press*, XCV., 21. Six stamp mills; 1,000 stamps; 4,000 tons a day. Saving by amalgamation 70–75 per cent. Each battery has 4 full-sized plates in series. Cyanide No. 1, capacity 1,800 tons a day, treats leachable portion of tailings from deep levels. Tails from oxidized ore treated in cyanide No. 2. 800 tons a day. Slime plant recently installed, Merrill filter press. See Ehle, *M. and Mins.*, XXVII., 358.

Dumping Residue at Kalgoorlie: von Bernewitz: *M. and S. P.*, XCV., 368. A great source of trouble. Conveyors are cheapest at 8 c. Bleichert, Electric Locomotive and Trucking with horses cost 10 to 12 c. a ton. See *M. and S. P.*, XCV., 459.

Slime Treatment: Recent Improvements in: D. J. Kelly. *M. and M.*, 28, 103. General remarks on the cyanide process and a description of the Kelly filter which consists of a pressure tank and a carrier supporting a series of filter diaphragms vertically disposed. Each diaphragm consists of a rectangular piece of double crimped wire cloth completely enveloped in a filter bag of cotton duck. Method of work given.

The Blaisdell Pressure Filter: M. and S. P., XCV., 188. Consists of a series of filter leaves, a pressure cylinder and pumps for vacuum, slime, water and solution. Capacity 500 tons slime a day.

The Fairchild Vacuum Filter: M. and S. P., XCV., 279. Suspended in a semicircular vat is a circular filter of 10 ft. dia. composed of 14 sections or groups of filters. Each section is composed of 10 filters radially attached to the inner periphery of the filter-wheel with suitable connections through the hollow spokes and hubs with a vacuum-pump on one side and a supply of compressed air on the other. Working automatic.

Large Tube Mills in the Cyanide Process: Electrochem. and Met. Ind., V., 286. Shows views of installation of 8 Abbé mills at the U. S. Reduction and Refining Co.'s plant at Colorado City. 5′ dia. 23′ long. Each mill 46 H. P., reduces tails from 12 to 100 mesh. 160 tons each per 24 hrs.

Tube Mills at Guanajuato: C. W. van Law. *M. and S. P.*, XCV., 205. A letter on the work done by Abbé Mills 4½′ x 20′ handling 80 tons pulp a day.

Chlorination of Gold-Ores: Laboratory-Tests: A. L. Sweetzer. *Bnll. A. I. M. E.*, 1907, 747. Gives methods and results of tests on a siliceous gold ore $25–26 Au and $0.68 Ag. Roasting. Chlorination in soda water bottles and agitated. Leaching. Barrel test 72 lb. charge.

The Clean-Up, Melting and Refining of Gold Bullion: G. W. Williams. *M. and S. P.*, XCV., 277. Abstracts from Monthly Journal of the Chamber of Mines of Western Australia, June, 1907. The gold slime and fine zinc are treated in three ways on the Rand. (1) Sulphuric Acid Method, (2) Sodium Bi-sulphate Method, (3) Taverner Process. The Bi-sulphate Method is described.

Electrolytic Processes for Refining Gold and Silver: Carl Hering. *Trans. Am. Elcetroch. Soc.*, XI., 415. U. S. Mint, Philadelphia. Crude bullion 30 Au 70 Ag is electrolyzed to obtain pure Ag. Residues electrolyzed to obtain pure Au. Platinum saved. Silver Process — anode 3 Au 7 Ag. Solution 3 per cent. silver nitrate: 1.5 per cent. Nitric acid. Current density 0.05 amp. per sq. inch. Gelatine gives firm deposit. See *Electrochem. and met. Ind.*, Aug., 1906, 306. Gold process. Wohlwill. Anodes moderately pure gold. Solution is chloride of gold strongly acidulated with hydrochloric acid. 50–55° C. Current density nearly 1 amp per sq. inch. (Solution worth $1,5000 a gal.) Gold comes out as adherent crystals.

Iron and Steel.

The Modern American Blast Furnace: B. Stoughton. *E. and M. J.*, 84, 145. Describes handling of raw material, loading the furnace, the blast furnace, blast and practice. Illustrations of Carrie furnaces, section of blast furnaces, blowing engines and gas engines at Lackawanna Steel Co.'s plant, etc. See

The Operation of the Iron Blast Furnace. *E. and M. J.*, 84, 307.

Heroult Electric Steel Process: Electrochem. and Met. Ind., V. 272. Illustrated article of a plant with Wellman open-hearth furnace and Heroult Electric Steel refining furnace. Output 60 tons daily. Capacity of O. H. 25 tons, of electric furnace 4 tons, run through in 1½ hrs.

The Electrothermic Reduction of Iron Ores: A. E. Greene and F. S. MacGregor. *Electrochem. and Met. Ind.*, V., 367. The points taken up were the design and construction of an experimental electric furnace of about 30 k. w. capacity, the measurement of the temperature of the molten charge, the factors affecting the temperature and the methods of regulating it, the effect of temperature and of composition of charge as the quality of iron produced, the calculation of the amount of electric energy required per ton of pig iron. Ores were iron sands from Pacific Coast and titaniferous iron ore from Essex Co., N. Y.

Electrical Smelting of Iron Ore: R. L. Phelps. *M. and S. Press*, XCV., 87. Describes the plant on the Pitt River, Shasta Co., Cal., for the reduction of local ores by Heroult Process and its trial run.

How may Quality of Steel Rails be Improved? H. M. Howe. *E. and M. J.*, 84, 21. Open hearth process, liberal use of aluminum or its equivalent, extremely slow casting, increasing discard.

Blow-holes in Steel Ingots: E. Von Maltitz. *Bull. A. J. M. E.*, 1907, 691–726. Deals with the subject of absorption of gases by iron and steel; hydrogen and nitrogen. Carbon monoxide and the formation of blow-holes. Prevention of overoxidation. Damage done by blow-holes in steel ingots.

Ternary and Quaternary Vanadium Steels: L. Guillet. *Revue de Métallurgie*, 1907, 775. Nickel vanadium steels most useful lie within the limits: — 0.1 to 0.3 per cent. °C, 2 to 7 per cent. Ni, 0.1 to 0.3 per cent. Va.

Boron Steels: L. Guillet. *Revue de Mét.*, 1907, 784. Consist of ferrite, pearlite and a boro-carbide of iron. Best results with 0.5 per cent. boron. When quenched the strength is very high without loss of ductility, compared with other special steels.

LEAD.

The Lead Roast-Reaction Process: Schenck and Rassbach. *Metallurgie*, 4, 455. The various systems of Pb, $PbSO_4$, PbS, SO_2 and PbO are worked out. The effect of concentration of SO_2 is very marked.

Lead Smelting in Utah: R. D. Brinsmade. *M. and M.*, 28, 216. Description of the U. S. Smelting, Refining and Mining Co.'s plant at Bingham Junction and the Murray plant of the A. S. and R. Co.

Lead and Copper Smelting at Salt Lake — I: W. R. Ingalls. *E. and M. J.*, 84, 527. Murray — Lead, 1,500 tons. Garfield — Copper, 1,250 tons. Bingham — Copper, 1,000 tons. U. S. — Copper, 1,000 tons, lead, 1,500 tons. Yampa — Copper, 500 tons. Highland Boy — Copper, 1,000 tons. Utah — Copper, 250 tons. Tintic — Lead, 300 tons daily capacity. U. S. Smelting Co., 6 copper furnaces and 5 converter stands. 15 l. h. reverberatories and 6 lead blast furnaces.

Lead and Copper Smelting at Salt Lake — II: W. R. Ingalls. *E. and M. J.*, 84, 575. Murray Plant. H. and H. process, 5 furnaces and 25 pots 9′ dia. 9 tons charge. Baghouse 4,160 bags 30′ long, 18 in. dia., 18 × 6 ft. fan 250,000 cu. ft. gas a min. Cost of baghouse, $150,000. Garfield works. 3 reverberatories and 2 blast furnaces. 4 converters. 16 McDougalls and 8 more being added. 25 H. and H. pots. The plant is shown by some excellent photographs.

Silver-Lead Smelting Practive — V.: T. S. Austin. *M. and S. Press*, XCV., 59. Roasting. Furnace Gases, 14–16 per cent. CO_2 and 8 per cent. CO by vol. on good reduction. Successful Running of Silver-Lead Blast Furnaces. Lead-Silver Smelting at El Paso.

The El Doctor Mines: T. D. Murphy. *M. and S. P.*, XCV., 241. Northeast part of Querétaro: Nearest railway point is Bernal. Have been worked since Spanish times. The mines are described. The smelter is a 36″ × 42-inch water jacket furnace, 40 tons a day. 12–14 oz. pressure. 1½–2 tons base bullion and 1½ tons matte a day. Bullion runs 8kg. silver, 6 gm. gold, 33 per cent. Sb and 62 per cent. Pb per ton. Matte hand-picked from slag and roasted in heaps. Bullion shipped to Germany. Slag 33 per cent. SiO_2. 28 per cent. FeO: 23 per cent. CaO: 10 per cent. Al_2O_3: 2·½ per cent. ZnO. Ag 60 gm.

Mining and Smelting at Granby, Mo.: E. T. Perkins. *E. and M. J.*, 388. Geology and mining discussed. Zinc ores are shipped to Kansas smelters. Lead ore smelted in 5 Jumbo furnaces. Charge 12,400 lbs. galena, 1,200 lbs. white fume, 400 lbs. blue, 1 bushel lime, 1,000 lbs. coal per shift 10 hrs. 85 per cent. of Pb is obtained: 2.5 per cent. slag running 40 per cent. Pb.

The Betts Process at Trail, B. C.: A. G. Wolf. *M. and M.*, 28, 11. Refinery of the Consolidated Mining and Smelting Co. of Canada, at Trail, B. C. Capacity 240 tanks, 70 tons refined lead a day. Base bullion runs 100 oz. Ag, 1 oz. Au, 0.2 per cent. As, 0.8 per cent. Sb, 0.25 per cent. Cu. Cast into anodes 300 lbs. each. Electrolyte a solution of lead in hydrofluosilicic acid (5 per cent. Pb. Silicon fluoride 9.3 per cent.) Slime treatment, drying. Smelt in small reverberatory with soda to 930–960 fine. Sulphuric acid parting. Plan of works given.

TIN.

Electrometallurgy of Tin: H. Mennicke. *Electrochem. Zeit.*, Mar.-Ap., May and June, 1907. Abs. *Electrochem. and Met. Ind.*, V., 468. Takes up detinning of scrap, tin boxes, etc. Review of patents on the electrometallurgy of tin for year.

Electrolytic Refining of Tin: Otto Steiner. *Electrochem. and Met. Ind.*, V., 309. The first attempt at electrolytic tin refining on a large scale was undertaken by Strauss & Co., in 1905, in their Penpoll Tin Smelting Works, Bootle. The Claus process was used, whereby tin alloys are electrolyzed in a 10 per cent. solution of sodium sulphide at 90° C. and ½ amp. per sq. decimeter electrode surface at less than 0.2 volts. Tin deposited on cathode and impurities (Pb, Sb, Cu, Fe, Bi, Ag, etc.) precipitated as sulphides, anode slimes. The method of working is given, cost of plant and working, etc.

ZINC.

The Electrometallurgy of Zinc and its Relation to Present Practice: W. McA. Johnson. *Trans. Am. Electrochem. Soc.*, XI., 265. Takes up Electromagnetic Separators, Wetherill, Dings, Cleveland-Knowles and the International. Electrostatic separators, Huff-Dolbear, Sutton-Steele-Sutton and the Blake-Morscher. Flotation processes are Potter, Delprat, Sanders and Elmore; "Tube concentration" of McQuestin etc. Igneous electrolytic processes — Swinborne-Ashcroft, for famous Broken Hill "middlings," chlorine blown through a small converter. Wet electrolytic processes. Haeffner process. Electric furnace propositions. Work of Snyder, De Laval and others.

Electrodeposition of Zinc: R. C. Snowdon. *Trans. A. Electrochem. Soc.*, XI., 121. With rotating cathodes. Good zinc deposits were obtained from acid or alkaline solutions even at 60 amp. per dm.^{2} Rate of rotation of cathode had enormous effect on upper limit of current density. Since zinc will precipitate readily from strongly alkaline solutions, the voltage across terminals can be made very low for any given current density. Increasing temperature or concentration increases size of the crystals. Presence of formaldehyde decreases size and so does increase of current density.

Zinc Smelting told by Photographs: M. and M., 28, 223. A general description of the American Zinc, Lead and Smelting Co's. plant at Caney, Kansas. Plant a subsidiary of the U. S. Smelting, Refining and Mining Co. Two roasters, Zellweger type 142 ft. × 15 ft.: roasts 70,000 lbs., dried ore 35 per cent. S in 24 hours. Six blocks of furnaces in pairs, each has 304 retorts (16 × 19). Manufacture of retorts. Illustrated by photos.

Zinc Smelting Works of Swansea, Wales: E. Walker. *E. and M. J.*, 84, 161. Output 30,000 tons per annum. Imports to Great Britain amount to 4 times that figure. Six plants, Vivian and Sons, at Morriston, English Crown Spelter Co., at Port Tennant, Dillwyn and Co., at Llansamlet, producing 180 and 150 tons a week each. Williams, Foster & Co.'s upper Bank plant 120 tons a week (the old Grenfell's works). The Villiers Co. and the Swansea Vale Spelter Co. produce about 30 tons a week each. Practically all custom work; Gas fixed Belgian furnace. English Crown plant typical. 13 furnaces, each 154 retorts, in two tiers. Siemens Regenerative type. Retorts 4 ft. × 11 × 7½ in. Clay a mixture of Stourbridge and Belgian. Mehler press. 12 weeks in drying: furnace. Smelting mixture is calcined calamine, roasted blende, waste and flue dust and anthracite. Prolongs of sheet iron used. Merton furnaces for blende.

The Treatment of Zinc Ores: F. W. Traphagen. *M. and M.*, 28, 5. Zinc-bearing ores in Rocky Mountain region in demand. Treatment by (1) Wet concentration, (2) Magnetic separation, (3) Electrostatic separation, (4) Oil concentration or flotation, (5) Pigment making, (6) Chemical methods, (7) Smelting for spelter.

Zinc Pigments, Manufacture of: E. W. Buskett. *M. and M.*, 28, 193. Modification of Wetherill process used at Coffeyville, Kansas. 60 tons ore a day. 2 five hearth McDougal's. 2 blocks of oxidizing furnaces each with 18 furnaces 6 ft. × 12 ft. Exhaust-fan to bag-room 90 ft. × 140 ft.

GENERAL.

The Temperature of Coke Ovens: E. and M. J., 84, 27. Semet — Solvay ovens, Bridgewater colleries, Lancashire. 1 p. m. to 10 p. m. rise 100° C., midnight 200° C., 1 a. m. 350° C., 2 a. m. 530° C., 3 a. m. 600° C., 4 a. m. 700° C., 5 a. m. 820° C., 6 a. m. 900° C., 7 a. m. 920° C., 11 a. m. 1,000° C., discharged. Thermocouple in exact center of charge.

A Granular Carbon Resistance Furnace: S. A. Tucker. *Trans. Am. Electrochem. Soc.*, XI., 307. Resistor crushed gas coke, 20-mesh, electrodes of Acheson graphite, graphite crucible.

Crucible Tongs: D. A. Johnson. *Met. Ind.*, V., 362. Uses of one-pronged, two-pronged and spade tongs given. Illustration of handy tongs for medium work.

Furnace for Melting Platinum: Met. Ind., V., 367. Describes a new crucible of very refractory material: is first heated by bunsen burner and then oxy-hydrogen flame played on platinum.

The Present Source and Uses of Vanadium: J. K. Smith. *Bull. A. I. M. E.*, 1907, 727. South American deposit. Calcined runs 58 per cent. vanadic oxide 25 per cent. silica. Vanadium melting point higher than platinum. Alloy of 2 iron to 1 vanadium melts at 1,375° C. and is the marketable form. Table of properties of vanadium steel compared with some other steels is given. 0.2 per cent. Va used. Erection of works at Pittsburgh, Pa., of American Vanadium Co., daily capacity of 2,000 lbs. Vanadium alloys settles question of supply.

Vanadium and its Effect on Steel and Copper Alloys: Met. Ind., V., 291. Products manufactured are: Hard ferro-vanadium for steel. Cupro-vanadium, ferro-nickel-vanadium for special steels. Also ferro-chrome-vanadium and ferro-tungsten-vanadium, aluminium-vanadium and nickel-vanadium (platinoid). Vanadium steel shows increased strength without diminishing ductility. Tables of physical tests are given.

Characters of Vibrations Accompanying Shock, Deduced from an Examination of Fractures: Ch. de Fréminville. *Rev. de Mét.*, 4, 433. First the fractures of glass are examined, then those of iron and steel, from which waves and curves of equal stress can be followed in the neighborhood of the point of impact and then further away.

Piping and Segregation, as Shown by Means of Wax Ingots: Influence of Conditions of Casting: H. M. Howe and B. Stoughton. *Bull. A. I. M. E.*, 1907, 561. Ingots of stearic acid with 1.5 per cent. green copper oleate, and a little cerasine. Photographs of sections of these ingots prove that the pipe is shortened and the segregate raised by slow casting, by casting with large end up, by retarding the cooling of the top: that the pipe is shortened by slow cooling.

German Silver for Flat Ware: The Manufacture of: T. Clare. *Metal Industry*, V., 3. Metal must be soft after annealing but strong after stamping and hammering. Best alloy runs: Ni 18, Cu 60, Zn 22. Melt the Cu and Ni and add zinc. Rolling and annealing mentioned. See "Casting German Silver for Rolling," *Metal Ind.*, August, 1905, and "Fire Cracks in German Silver," *Metal Ind.*, March, 1905.

Manganese-Zinc: A. E. Hobson. *Metal Ind.*, V., 7. Used for manganese-copper as a deoxidizer in manufacture of brass, etc. Mn 15, Zn 85. Used in the proportion of 1 per cent. Manganese tin is also used.

Lead Alloys: Lincoln Burrows. *Metal Industry*, V., 9. Gives composition of a few lead alloys with melting points in degrees F.

The Working-Up of Metal Turnings: A. Saxer. *Metallurgie*, 4, 517. An intimate mixture of white metal and bronze was unable to be separated mechanically. Liquation only partly successful. Borcher's converter used and bronze produced.

The Electrolytic Deposition of Nickel Zinc Alloys. E. P. Schoch and A. Hirsh. I. *J. Am. Chem Soc.*, 1907, 314; II. *Trans. A. El. Soc.*, XI., 135. When a solution which contains both nickel and zinc salts is electrolyzed the metal deposit obtained is always an alloy of both

metals. Mixtures of sulphates used: ratio of Zn to Ni in an alloy is always 4.5 to 14 times their ratio in the electrolyte, depending on total concentrations in latter of the two salts. Deposition of alloys high in nickel on a mercury cathode. Potentials of alloys against zinc in N/1 $ZnSO_4$ solution. Determination of equilibrium conditions of zinc, nickel and their alloys with solution mixtures of their sulphates.

New Alloys: Met. Ind., V., 184. From Metallurgie. Allen's metal: Cu 55.25, Pb 44.61, Sn 0.14. Nickel aluminum: Al 74.4, Zn 19.87, Cu 3.49, Mn 1.42, Fe 0.45, Si 0.37.

Influence of Various Metals on the Working Qualities of Alloys: Met. Ind., V., 312. Abs. from Der Metallarbeiter. German silver for rolling: (1) Zn 20, Cu 60, Ni 25; (2) Zn 25, Cu 50, Ni 25; (3) Zn 26, Cu 60, Ni 15, has yellow color; (4) Zn 48, Cu 60, Ni 15, very fusible. For casting: (5) Zn 23.7, Cu 55.8, Ni 13.4, Pb 3.54, Sn 4; (6) Zn 5.6, Cu 69.9, Ni 19.8, Cd 4.7; (7) Zn 19.5, Cu 65.2, Ni 13, Ag 2.05, Fe 0.12. Bearing metals in use in Germany: (1) Sn 83, Sb 11, Cu 6; (2) Sn 42, Sb 16, Cu 2, Pb 40.

Cleaning and Pickling Copper-Alloy Castings: Metal Ind., V., 22. Dilute sulphuric acid used. 8–10 per cent. commercial acid to 100 parts water, at 140–160° F. Time 20–30 mins. Matte surface obtained and color improved by following with a short immersion in a mixture of commercial sulphuric and nitric acids, to which a little sodium chloride and lamp black have been added. Rinse and dry. To prevent stain dip in dilute solution of argol or crude potassium bitartrate.

Metallurgical Calculations: J. W. Richards. *Electrochem. and Met. Ind.*, V., 266. Smelting of Copper Ores. V., 304 — Pyritic Smelting. V., 352 — "Bringing Forward of Copper Matte, Bessemerizing of Copper Matte." V., 399 — The Electrometallurgy of Copper. V., 458 — Electrolytic Refining of Impure Copper.

Old Sheffield Plate Trade Marks: L. E. Taylor. *Metal Ind.*, V., 267. Several old marks are given. Old Sheffield plate was stamped with the leopard's head. Some specimens have silver edges to conceal the soldering on the mounts, also a silver shield to enable buyer to have his own crest engraved on it.

Sheffield Plate and Electro-Plate: L. E. Taylor. *Met. Ind.*, V., 364. Old Sheffield plate was really a sham, for the base metal was covered or coated with silver, by soldering. Close plating consisted of tinning iron or brass and then soldering over with standard silver sheet. In 1838 electro-plating was introduced and soon superseded Sheffield plating.

British Hall-marking: J. Horton. *Met. Ind.*, V., 107. Hall-marking dates back at least 600 years. There are now seven assay offices — London, Birmingham, Chester, Sheffield, Edinburgh, Glasgow and Dublin — each having its own mark. Every manufacturer must register his name and mark, usually initials. The hall-marks affixed consist of the maker's mark, the local mark, a variable date letter. Before the abolition of duty a further mark indicated payment of duty and was a Queen's Head. If the article is standard silver the mark is a Lion

Passant Gardant. Britannia silver, a new standard in 1697, of 959 fineness, was marked with the figure of Britannia. Gold standard is 22 carat and is stamped with the crown and 22.

British Hall Marks: Met. Ind., V., 365. Shows hall marks for foreign goods.

Early "Cir Perdue" Work in France and its History in Our Own Country: J. G. Niederer. *Metal Ind.*, V., 195. Mentions the work of Honore Gonon and his son. See V., 295, 236.

Early Irish Art Metal Work: Metal Ind., V., 227. Article with several illustrations of Irish work.

METALLOGRAPHY — PYROMETRY.

On the Estimation of High Temperatures: G. K. Burgess. *Trans. Am. Electrochem. Soc.*, XI., 247. Temperature scale up to 1200° C. is that of the gas thermometer. Fixed points: Sn 232; Kahlbaum Zn 419; Pittsburg Red. Co. Al 658; Kahlbaum Sb 630.5; Cu 1084. Above 1200° C. there is no generally accepted scale and extrapolation in terms of some phenomenon varying with the temperature must be resorted to. For thermo couples using the formula $E = a + bt + ct^2$ the melting point of Ni = 1427, Pd 1530–1535, Pt 1706–1710. Using Holman's formula $E = mt^n$ the palladium and platinum melting points become 1543 and 1731° respectively. By radiation methods Pt = 1746–1753° C. Platinum ribbon method gave Mp of Fe 1505°, Cr 1482, Co 1464, Ni 1435, Mn 1207° C.

Electric Tube Furnace for High Temperature Instruments: S. A. Tucker. *Trans. Am. Electrochem. Soc*, XI., 303. Agglomerated carbon tubes made by National Carbon Co. of Cleveland were used for furnace, packed in petroleum coke in firebrick receptacle.

Melting Points: J. K. Clement. *Physical Review*, June, 1907. Abs. *Electrochem. and Met. Ind.*, V., 321. New measurements with gas thermometer. Zn 418.4° greatest deviation from the mean ±0.4°, Ag 957.5°, Au 1,059.1°, Cu 1,080.2°, greatest deviation from the mean ±0.5°.

Melting Point of Tungsten: H. von. Wartenberg. *Ber.*, 40, 3287. Lies between 2,800° and 2,850° C.

Radiation from and Melting Points of Palladium and Platinum: Waidner and Burgess. *Reprint 55, Bull. Bureau of Standards*, Vol. 3, No. 2. Melting point of palladium is 1,546° and that of platinum 1,753° C.

Melting Points of the Iron Group Elements by a New Radiation Method: G. K. Burgess. *Reprint No. 62, Bull. Bur. of Standards.* A minute quantity of metal is melted on a Pt strip by electrically heating the latter. Melting point is observed by microscope and optical pyrometer. Iron 1,505°, Cr 1,489°, Co 1,464, Ni 1,435, Mn 1,207. Correct to within less than 10° C.

The Work of Cartaud on the Passage from the Liquid to the Solid State: F. Osmond. *Rev. de Mét.*, 4, 819. The cellular structure of metals and its evolution, illustrated by 72 microphotographs.

Research on the Constitution of Alloys by Tammann's Method of Thermal Analysis: A. Portevin. *Rev. de Mét.*, 4, 797. Deals with Tammann's method of "Thermal Analysis," (1) for a eutectic; (2) point of transition; (3) allotropic transformation; (4) two conjugate solutions. A table of alloys already worked out is given.

The Alloys of Antimony with Manganese Chromium, Silicon and Tin; of Bismuth with Chromium and Silicon; of Manganese with Tin and Lead: R. S. Williams. *Zeits. für Anorg. Chemie*, 55, 1. Antimony and manganese form a compound $SbMn_2$. $SbMn_2$ and Mn (919 and 1,228° C.) form a eutectic at 900° C. and 55 per cent. Mn. $SbMn_2$ transforms at 250° C. Between 0 and 40 per cent. Mn the alloys consist of Sb or Sb_2Mn_3 in the eutectic (9 per cent. Mn 577° C.). The freezing point curve falls from the maximum at $SbMn_2$ to the eutectic 9 per cent. Mn, with a reaction point at 39 per cent. Mn 852° C., where Mn_3Sb_2 forms. This holds some Sb in solid solution, so that the eutectic runs only from 0 to 32 per cent. Mn. Sb_2Mn_3 transforms pure β to α form at 320° C.

Antimony and chromium form a compound SbCr, a maximum at 30 per cent. Cr 1,111° C. This forms a eutectic with Cr (1,553° C.) at 38 per cent. Cr 1,100° C. Sb_2Cr and Sb form a eutectic at 2 per cent. Cr and 620° C. The curve from the eutectic point to the maximum at SbCr shows a reaction point at 5 per cent. Cr and 675° C. with the formation of Sb_2Cr.

Antimony and tin show a freezing point of three branches all sloping toward the melting point of tin and meeting at 50 and 10 per cent. Sb (420 and 243° C.). Four solid solutions are formed (*a*) 90 per cent. Sb, (*b*) 53 per cent., (*c*) 50 per cent. and (*d*) 10 per cent.

The curve for antimony and silicon shows a constant temperature 630° C. for a lower or solidus curve. The liquidus slopes slightly from pure Si 1,414° C. to 90 per cent. Sb alloy 1,248° C. and then takes a sharp drop to the freezing point of Sb. Silicon crystallizes out in a groundmass of antimony.

Bismuth and silicon do not alloy to any extent, neither do bismuth and chromium.

Manganese and Tin. The curve slopes from the freezing point of manganese 1,228° C. to that of tin and shows three reaction points: at 65 per cent. Mn 989° C. where Mn (10 per cent. Sn in solution) and the solution react to form Mn_4Sn: at 45 per cent. Mn 896° C. where $SnMn_4$ and the liquid form $SnMn_2$: at 8 per cent. and 548° where SnMn is formed. The solidus up to 32 per cent. Mn is at 231° C., the freezing point of pure tin.

Manganese and Lead. The critical point curve and the freezing point curve cut at 90 per cent. and about 10 per cent. Mn at 1,197° C. The lower points are those of pure lead 326° C.

Antimony Lead Alloys: Gontermann: *Zeits. für Anorg. Chemie*, 55, 419. Curve consists of two branches from the freezing points of Sb. 630 and lead 326 cutting at the eutectic 247° C. and 13 per cent. Sb.

Aluminium Bronzes: The Constitution of. B. E. Curry. *J. Phys. Chem.* XI., 425. Gives the freezing point curve and equilibrium dia-

gram above 400° C. Seven phases occur, α, 100 — 91 per cent. Cu. β 90 — 85 per cent. Cu at 1,000°, γ 84 — 79 per cent. Cu. δ 77 — 75 per cent. Cu at 700° C. ϵ 76 — 73.5 per cent. Cu. $CuAl_3$ 53 — 7 per cent. Cu: η 0 — 11 per cent. Cu. η and $CuAl_2$ (585° C.) form an entectic at 32½ per cent. Cu and 543° C. From $CuAl_2$ to 87.5 per cent. Cu (a maximum) the curve shows three reaction points, 585 at 53.7 per cent. Cu, 620° at 60 per cent. Cu, 846° C. at 72 per cent. Cu. The solid solution β breaks up into α or γ + eutectoid at 566° C. The δ phase is also unstable and breaks down into ϵ, etc.

The Tensile Strengths of the Copper Aluminium Alloys: B. E. Currey and S. H. Woods. *J. Phys. Chem.*, XI., 461. Various copper-rich and aluminium-rich alloys were made and annealed at definite temperatures with regard to the equilibrium diagram. In the aluminium-rich series the maximum dependable strength occurs about 10 per cent. Cu, 28,000 lbs. per sq. inch. With 90 per cent. Cu, chilling or annealing above 566° gives a strength of 100,000 lbs. per sq. inch.

The Electrolytic Corrosion of Brasses: A. T. Lincoln, D. Klein, P. E. Howe. *J. Phy. Chem.*, XI., 501. A series of copper zinc alloys were annealed at 400° C. to bring them to equilibrium and then subjected to electrolytic corrosion in normal solutions of some of the more common sodium and ammonium salts. For the brasses with 50 per cent. Cu up the corrosion product is of the same composition as the test piece. In the low copper brasses the corrosion product is practically pure zinc.

Bismuth-Thallium Alloys: M. Chikashigé. *Zeits. für Anorg. Chemie*, 51, 328: A compound Bi_3Tl_2 (37 per cent. Tl 212° C.) forms a simple series of alloys with Bi, with a eutectic at 20 per cent. Tl and 195° C. It also forms a simple series with a solid solution 89 per cent. Tl (303.5° C.) forming a eutectic at 53 per cent. Tl and 186° C. This eutectic however only runs from 39 to 67 per cent. Tl. Between 89 per cent. Tl and 100 per cent., a series of solid solutions occur showing a minimum at 93 per cent. Tl. Transformations in the solid occur between 60 and 75 per cent. Tl with the formation of $BiTl_3$ + Bi.

Copper-Bismuth Alloys: Jeriomin. *Zeits. für Anorg. Chemie*, 55, 412. The curve shows a moderate slope from the freezing point of Cu to 10 per cent. Cu and then falls sharply to the freezing point of Bi. Copper freezes out in a groundmass of bismuth. Each holds ½ per cent. of the other in solid solution.

The Alloys of Copper with Nickel and Gold: the Electrical Conductivity of Solid Metal Solutions: Kurnakow and Żemczużny. *Zeits. für Anorg. Chemie*, 54, 149. The freezing point curve of Ni and Cu is a slightly convex line joining the two freezing points whilst the conductivity curve shows a sharp drop from Cu to the 10 per cent. Ni alloy whose conductivity is slightly below that of nickel. Between 10 and 100 per cent. Ni the minimum is slight at 40–60 per cent. The gold-silver and gold copper diagrams are given.

Silicon-Copper: M. Philps. *Metallurgie*, 4, 587, 613. Alloy Cu_7Si_2 formed. Chief use is in deoxidizing of copper, bronzes, etc.

Iron Molybdenum Alloys: Lautsch and Tammann. *Zeits. für Anorg. Chemie*, 386. The addition of Mo to Fe reduces the freezing point from 1,525° C. to 1,354° at 42.5 per cent. Mo above which the freezing point rises again to 1,487° C. at 60 per cent. Mo. Lower points were found in the solid. The constitution is explained by supposing that iron and molybdenum form a compound X and we are dealing with a ternary system of the three.

Iron Chromium Alloys: Treitschke and Tammann. *Zeits. für Anorg. Chemie*, 55, 402. The freezing points lie on an irregular curve joining the freezing points of the two metals. The cooling curves are given for each 10 per cent. step. A compound of iron and chromium, X, is supposed to be formed and we are dealing with the ternary system iron, X, chromium.

Iron to Lead, Bismuth, Thallium, and Cadmium: On the Behavior of: Isaac and Tammann. *Zeits. für anorg. Chemie*, 55, 58. No alloys are formed.

The Iron-Platinum Alloys: Isaac and Tammann. *Zeits. für anorg. Chemie*, 55, 63. The freezing point curve is a concave slope joining the two freezing points and a series of solid solutions γ are formed. At lower temperature, changes in the solid occur. Between 0 and 50 per cent. Pt the change point falls from 800 to 350° C.: between 60 and 90 per cent. Pt there are two transformations.

Iron-Tungsten System: H. Harkort. *Metallurgie* 4, 617, 639, 673. The method of work given at length. The freezing point curve very irregular, 1.23 per cent. W. 1,442°, 1.78 per cent. W. 1,504°, 1.96 per cent. W. 1,467°, 2.10 per cent. W. 1,438°. The critical ranges on heating and cooling of alloys up to 19.6 per cent. W. worked out. Change from β to α iron 740–785° C. γ to β iron 863–941° C.

A New Iron-Carbon Phase, Osmondite: H. M. Howe. *Electro-Chem. and Met. Ind.*, V., 347. Heyn and Bauer discovered osmondite, which appears to be a solid solution of carbon in normal or alpha iron. Austenite is such a solution in allotropic gamma iron. The paper discusses Heyn and Bauer's work and tabulates the different transformations between austenite and pearlite.

Alloys of Nickel and Lead: A. Portevin. *Rev. de Mét.*, 4, 814. Ni 1,484° and Pb 327° C. Separate into two layers for the critical curve cuts the freezing point curve at 1,365° and at 7 per cent. and 60 per cent. Ni. The eutectic 0.07 per cent. Ni freezes at 323° C.

Palladium-Gold Alloys: R. Ruer. *Zeits. für anorg. Chemie*, 51, 391. Curve similar to that of Pd-Ag but is much steeper between 0 and 30 per cent. Pd.

Palladium-Copper Alloys: R. Ruer. *Zeits. für anorg. Chemie*, 51, 223. The curve is a concave slope between the freezing points of the two metals, and the alloys consist of a series of solid solutions.

Palladium-Silver Alloys: R. Ruer. *Zeits. für anorg. Chemie*, 51, 315. The freezing point curve forms a convex slope joining the two freezing points, palladium 1,541 and silver 961.5° C.

Platinum Alloys: Doerinckel. *Zeits. für anorg. Chemie*, 54, 333. Platinum-copper alloys give a freezing point curve slightly concave, forming a series of solid solutions from Au 1,080 to Pt 1,744° C.

Platinum and silver give a curve with two branches both slightly convex, meeting at 32 per cent. Pt and 1,185° C. which is the critical temperature up to 80 per cent. Pt +, the solid forming having the composition $PtAg_2$.

The platinum gold curve is a convex slope between the freezing points of the two metals and the space between the solidus and liquidus is very wide compared with that of the PtCu series.

Platinum and tin show a maximum at 1,281° C., 62.5 per cent. Pt or PtSn. Between this point and pure tin the curve falls in three branches, giving two critical points, the first at 845° C. and 35 per cent. Pt, corresponding to formation of Pt_3Sn_8, the second at 535° C. and 9 per cent. Pt, corresponding to the formation of Pt_3Sn_8. PtSn and Pt_3Sn form a eutectic at 1,080° C. and 72 per cent. Pt. The curve from this eutectic to the freezing point of Pt 1,744° C. is a continuous slope with a critical point at 1,365° C. and 80 per cent. Pt corresponding to the formation of Pt_3Sn. Pt_2Sn_3 changes from the α to the β modification in heating above 730° C.

Platinum and lead form a curve with a eutectic at 5 per cent. Pt 290° C. : from this point to the freezing point of platinum the curve shows three critical points at 10 per cent. Pt and 360° C., at 45 per cent. Pt and 790° C. (PtPb) at 60 per cent. Pt and 910° C.

Zinc-Cadmium Alloys: Hindrichs. *Zeits. für anorg. Chemie*, 55, 415. The freezing point curve consists of two inclined branches from the melting points of the two metals 419 and 322° C., cutting at the eutectic point 270° C. and 82.6 per cent. cadmium.

The Ternary Alloys of Lead, Magnesium and Tin: Vegesack. *Zeits. für anorg. Chemie*, 54, 367. The three binary curves are given. Mg_2Sn forms eutectics with Mg and Sn. Mg_2Pb forms eutectics with Mg and Pb. Pb and Sn form a single eutectic. The ternary diagram is given in three dimensions.

The Freezing-point Curves of the Binary Systems Galena-Iron Pyrites and Galena-Argentite: Friedrich. *Metallurgie*, 4, 460. PbS (1,114° C.) and FeS (1,187° C.) form a eutectic at 70 per cent. PbS and 863° C. PbS and Ag_2S (835° C.) form a 77 per cent. Ag_2S at 630° C.

COLUMBIA CHAPTER SOCIETY OF THE SIGMA XI.

The first regular meeting of the society was held October 31 with the Department of Physics and a paper presented by Mr. L. B. Morse on "The Selective Reflection Characteristic of Salts of Carbonic and Other Oxygen Acids," an abstract of which follows.

I. *The Selective Reflection of Carbonates as a Function of the Atomic Weight of the Base.*

Polished plane surfaces of (Mg, Ca, Mn, Zn, Sr, Ba, and Pb) CO_3 were prepared and the ratio of the reflected to the incident radiation was measured at short wave-length intervals between 4 μ and 15 μ. The following are the principal conclusions reached:

1. The reflection curves for all the carbonates examined show between 4 μ and 15 μ three, and only three, bands of abnormal reflection. Abnormal reflection interpreted means a free resonance period of the molecule.

2. The bands fall into three separate and definite spectral regions, which are distinct from the regions where the salts of other acids, so far as known, show reflection maxima.

3. With few exceptions, an increase in the atomic weight of the base causes a shift of all three reflective maxima toward long waves by an amount roughly proportional to the change in atomic weight of the base.

II. *The Rôle Played by Oxygen in the Selective Reflection of Carbonates, Nitrates, Sulphates and Silicates.*

Combining with the data on carbonates, the scattered observations of other observers on nitrates, sulphates, and silicates, the tentative hypothesis has been made that the oxygen atom is the one chiefly responsible for the marked reflection observed.

The wave-lengths of the first reflection bands in $CaCO_3$, KNO_3,* $CaSO_4$ and $MgSiO_3$† are platted as abscissae, and as ordinates the weights of the acid forming elements combined with O_3 (O = 12, N = 14, $\frac{8}{4}$S = 24, and Si = 28). The lines show clearly that a small increase in the weight of the acid-forming element produces a much greater displacement of the reflection band than does the same increase in the weight of the base, and this is in full agreement with the chemist's view of the relative strength of the bands existing between the *acid-forming element* and oxygen, and that between the *base* and oxygen. The results suggest a new and far-reaching method by which it may some time be possible to express the dynamical relations existing between the separate atoms of a molecule, and thus the present conception of chemical bands and linkages be given a broader significance.

* Two values are platted for KNO_3 corresponding to the results obtained by two independent observers, Pfund and Coblentz.

† If a correction be applied to correct for Mg being lighter than Ca this would bring the $MgSiO_3$ point even nearer the line drawn.

At the second meeting held November 12, with the Department of Chemistry, Professor C. F. Chandler addressed the members on "Recent Progress in Electrochemistry." The subject matter dealt with the commercial application of electricity to produce chemical reactions as practiced at some of the large manufacturing plants in the Niagara Falls district. Professor Chandler has had unusual opportunities to study the problem and presented it in his characteristically interesting style.

C. E. M.

BOOK REVIEWS.

A Treatise on Hydraulics. By WILLIAM CAWTHORNE UNWIN, LL.D., F.R.S. Adam and Charles Black, London, 1907. 8vo, pp. xi + 324, illustrated. Price $4.25 net.

The concise manner of presentation, and the lucid explanations and solutions of the various problems and theorems, all bear evidence of the author's considerable experience as a teacher. The consistent employment of Bernouilli's Theorem throughout the work in the establishment of other theorems of practical import is very satisfactory; the inclusion of two chapters dealing with the statics, dynamics, and flow of compressible fluids is commendable; and the presentation of numerous problems for numerical solution will certainly appeal to the teacher.

On the other hand, some features of a minor nature may justly be criticised without detracting from the general excellence of the work; for example, the author merely states formula (12), page 75, without suggesting its rational derivation, or indicating its origin, although it is founded on a well-known formula established by Daniel Bernouilli, which gave rise to considerable criticism and dispute, and was ably discussed, together with its criticisms, by Julius Weisbach in the Allgemeinen Maschinen-Encyclopædie. Also, in view of the practical importance of the work, too scant consideration is given, in the writer's estimation, to the extensive investigations on the flow of water over broad-crested weirs carried on at the Hydraulic Laboratory of Cornell University in 1898–99. Again, there is no mention of the backwater and drop-down surface curves, though both, on occasions, are of considerable practical importance.

On the whole, it is a work which should appeal to the more earnest class of students, to teachers of hydraulics and allied subjects, and to practicing engineers.

A. B.

Text-book on Hydraulics, Including an Outline of the Theory of Turbines. By L. M. HOSKINS. Henry Holt and Company, New York, 1907. 8vo, cloth, pp. iv + 266. Price $2.50.

In this work the author develops in simple and satisfactory manner the fundamental and extremely useful theorem of Bernouilli, or, as it is sometimes called, the general equation of energy in steady flow; and from this basic relation he derives many of the hydraulic formulae which in text-books are often developed independently, frequently with a gain in simplicity or brevity, to be sure, but with the sacrifice of uniformity of method.

In the portion of the work devoted to the treatment of the laws of fluid friction, some of the principal results of the investigations of hydraulicians on this important topic are brought together very briefly

and concisely. About three pages are devoted to a meagre and inadequate treatment of the flow of wells. Under the topic Non-uniform Flow, the case of backwater is treated, but the drop-down surface curve is not considered. In the chapter devoted to gauging the rate of discharge, too scant consideration is given to the method of measurement by means of floats, and nothing is said regarding the numerical coefficients for reducing particular observed velocities to mean velocities. The current meter, so important an instrument in certain kinds of hydraulic investigations, also receives very scant notice, much less than is accorded the Pitot tube ; and under the latter heading Freeman's interesting employment of this device in determining the variation of velocities throughout the cross-sectional area of jets from nozzles is not mentioned, although these experiments established the fact that the Pitot tube is a valuable instrument for the measurement of high velocities. The case of discharge from nozzles is not treated. In Article 82, the author states that the loss of head due to sudden enlargement of section " is found by experiment " to be given approximately by the familiar formula (which he states) for such case, instead of deriving it, or indicating how it may be derived. The elements of the theory of turbines and water-wheels is briefly presented, and the general equation of energy as developed for an incompressible fluid is modified so as to apply to the case of compressible fluids ; in the latter case, however, the flow through pipes is not considered. The important considerations of the discharge over broad-crested weirs, recent and reliable knowledge of which is so largely dependent on the investigations of Bazin in France, and on the Cornell University experiments in this country, which latter may be termed an extension of Bazin's work to the cases of higher heads and full-sized dams of various cross-sections, the author passes over with the remark that " no precise estimate can be made of the rate of discharge " for such cases.

There are many minor omissions, and it is the opinion of the writer that the author attempted to cover too large a field in a limited number of pages, with the result that he was compelled, in many instances, to make general statements where specific information would have been of more value.

A. B.

A Manual of Fire Assaying. By CHARLES H. FULTON, President, South Dako a School of Mines. New York, Hill Publishing Co., 1907. Pp. xii + 178. Price $2.00.

In this work Mr. Fulton has made a commendable attempt at the impossible, *i. e.*, to put fire assaying, with our present lack of exact knowledge, on a scientific basis — although he has made a long stride forward.

This book has considerable merit and is composed of treatises on the the following subjects.

Chapter 1 contains useful information on different types and styles of assay furnaces, with data on fuel consumption. Also describes various tools and apparatus.

Chapter 2 describes various fluxes and reagents with their use and reactions and includes necessary tests of same.

Chapter 3 gives a fairly good description of ore sampling, but is deficient on sampling of base bullions.

Chapter 4, balances and weights. This subject is given more space than is customary in assay books and it has received the attention its importance warrants.

Chapter 5, reduction and oxidation. One of the most important fundamental points in assaying is excellently treated.

Chapter 6. In the description of the crucible assay and assay slags, Mr. Fulton has departed from the regulation cook-book style and shows the reasons why. This is one of the strong points in the book.

Chapter 7 consists of an excellent treatise on cupellation, giving useful information relating to cupels, and a good description of the process, but is marred by a few questionable statements. The assertion is made that PbO formed between 700° and 750° C. in cupellation is solid. Is it possible Mr. Fulton would suggest that a sponge will absorb water in the form of ice? The inclusion of a table of experimental results by T. K. Rose is superfluous and deterimental, as it does not represent assay conditions and results, and therefore is misleading.

Chapter 8, parting. A knowledge of information contained herein should prevent any difficulties on this point.

Chapter 9. This chapter contains a large amount of useful and valuable matter on the assay of ores containing impurities, such as sulphides, arsenides, copper, zinc, etc. Also various metallurgical products.

Chapter 10, special methods of assay. This is practically a continuation of the previous chapter, treating of less common material. It contains a forceful object lesson in a table of results from telluride assays.

Chapter 11 is composed of much data on the loss of gold and silver and the retention of other metals by the bead.

Chapter 12 describes the Bullion Assay methods in general use.

Chapter 13 discusses many points relating to assay of ores and alloys of platinum, iridium, gold and silver, and gives two methods for final separation. If the first part of the first method had been combined with the last part of second method, better results would be obtained, at least when small amounts are considered.

Chapter 14. The determination of tin, mercury, lead, bismuth and antimony by the fire method is briefly considered in this chapter. But they have received all the space that their use and accuracy warrants.

The book throughout is perfectly reliable in its main features, but it is surprising that the assay of cyanide solutions should have been omitted. Only in small matters has the author gone astray, which may be illustrated by the following examples. The statement is made that Na_2CO_3 is decomposed by high temperatures. This statement is ambiguous and incorrect in the connection used. Sodium carbonate is not decomposed by heat alone at assay temperatures. The statement that antimony makes lead buttons hard was probably copied, as antimonal lead is hard lead. Riders and small weights are made of aluminum, not platinum, as stated.

In conclusion Mr. Fulton is to be congratulated in filling a long-felt

want, by producing a work that can be satisfactorily used as a text-book. The adoption of his book by a number of universities is an appreciative testimony to his efforts. The writer has introduced it at Columbia.

E. J. HALL.

Electro-Analysis. By EDGAR F. SMITH. Fourth edition revised and enlarged. P. Blakiston's Son & Co., Philadelphia, 1907. 12mo. Limp leather, pp. 336. Price $2.50 net.

The excellent work of Professor Smith and his co-laborers on electrochemical analysis is familiar to all, and the appearance of this new edition, which contains all of the most recent work in this important field, should receive a hearty welcome.

The author has eliminated from this new edition nearly all of the space devoted, in previous editions, to "Sources of Current," for, as stated in the preface: "information about them is best obtained from publications devoted especially to them." An interesting section of ten pages has been added on "Theoretical Considerations," in which the theories of electrolysis are briefly, but concisely stated. Twenty-one pages are devoted to "The Rapid Precipitation of Metals" and the "Use of a Mercury Cathode." The subjects are treated historically and numerous illustrations show the forms of apparatus used.

Following the plan of previous editions, the metals are first taken up each in a separate section, and following this the separations are considered. In addition to the conditions for the electrolysis and separations of metals with the use of stationary electrodes, directions, forms of apparatus, etc., are also given for determinations by means of a rotating anode, also by means of a rotating anode and mercury cathode, where such have been applied.

Another new section (eleven pages) is devoted to the separation of metals by graded potential.

Perhaps the most interesting portion of this volume is the section (thirty-four pages) on the determination of anions, and the simultaneous determination of anions and cations, and numerous separations of the alkali and alkaline earth metals, a field in which so much good work has been done by the author.

The final section treats of the "Combustion of Organic Compounds'" in which the electric current is employed.

The reviewer has no hesitation in recommending the book as one of the best on the subject and that it should be in the hands of all chemists interested in analytical or electrochemical work.

The publisher has done justice to the subject matter in producing a book of first quality.

F. J. METZGER.

Standard Polyphase Apparatus and Systems. Fifth Edition. By M. A. OUDIN. New York, D. Van Nostrand Co. 369 pages, 207 illustrations. Price, $3.00.

The first chapter of the book is devoted to a very brief theoretical discussion of the phenomena which occur in alternating current cir-

cuits. However, in the following chapters the author describes very fully the features of construction and the characteristics of operation of alternating current apparatus. The matter is very well presented and includes a great deal of valuable data, and on account of this the book is of considerable worth, and becomes, without doubt, of great assistance to engineering students.

M. A.

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Journal of Applied Science

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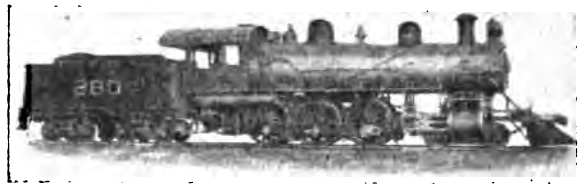

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THE

SCHOOL OF MINES QUARTERLY.

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SAMPLING OF MINE DUMPS.

BY HENRY S. MUNROE.

It frequently happens that mine dumps contain low grade ore, originally rejected as too poor for treatment, which later, under more favorable conditions, may be worked with profit. The old waste dumps may thus constitute an important asset of the mine, the value of which must be accurately determined. The sampling of such waste dumps presents a difficult problem. Portions of the dumps which have accumulated during periods of active development; for example, during the sinking of a shaft or the driving of a tunnel, may be absolutely barren; while other portions representing the growth during periods of wasteful or careless management, may be very rich. In any case the distribution of values is likely to be very irregular. If the dumps be formed in the ordinary way, by dumping cars at the top of the bank, the stratification, and therefore the zones of rich and poor material, will be in inclined layers. Moreover, as the cars are dumped at the top, the larger pieces tend to roll to the bottom of the slope, while the fine stuff lodges and remains near the top. The upper portion of the dump, as containing most of the fine material, is therefore likely to be rich; and the bottom, consisting of lumps and coarse material, is in general poorer than the average.

Figs. 1 and 2 are intended to illustrate this sizing effect, which has been purposely exaggerated in the drawings. In practice many large lumps will remain in the upper part of the dump and some fine stuff will be carried down the slope. The outside of the dump

represents in most cases the later additions only, and the material deposited in earlier years is deeply buried. Finally, the dumps will contain much ore and rock in very large lumps and a due regard to

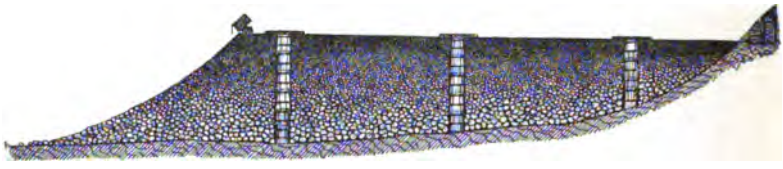


FIG. 1. Longitudinal section of a mine dump with shafts sunk for sampling.

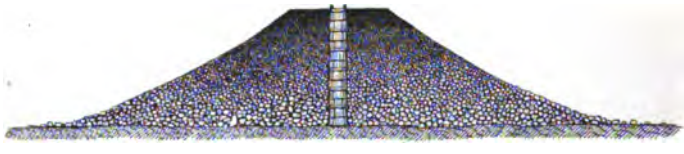


FIG. 2. Cross section of a mine dump with sampling shaft.

the effect of such large pieces on the value of the sample taken, will necessitate handling large quantities of material. For example, if we adopt the inch-ton sampling scale,* the presence of ten inch pieces would indicate one hundred tons as the minimum total weight of samples to be taken. Or even if we take a two-and-a-half-inch-one-ton sampling scale, the presence of many lumps 25 inches in diameter will lead to the same result.

The usual method of sampling a dump is to take numerous small samples from the sides and top, which are either assayed separately or united to make one or more large samples. If this method be followed the points to be sampled should be determined by measurement, either at the corner of 25-foot squares, or better, in some cases, located on contour lines at uniform distances apart. In either case the samples should be given weight in the final average in proportion to the amount of ore that they may represent in the original pile, bearing in mind the probable concentration in values at and near the top of the dump. If the samples be located on contour lines, those on each contour may be combined and averaged separately, and the value for each contour averaged on the basis of the horizontal dump area represented.

* Richard's "Ore Dressing," p. 852.

In dump sampling, on account of the large amount of material to be worked down, a small crushing plant is essential. If the crushing be done by hand, with hammers and by mortar and pestle, a large force of men is necessary and the effective protection of the samples is made difficult. The crushing machines should be somewhat larger than the usual laboratory size and should have a capacity of several tons per day. The crushing plant may be installed at some central point where power is available and where the necessary counter shafting can be erected. Such a plant will be expensive, but in an important examination may save its cost several times over.

Last year the writer had to determine accurately the value of some very large dumps at a mine in Bolivia. In order to avoid the sources of error incident to sampling in the ordinary way, and to reach the deeply buried portion of the dumps, which were believed to be rich, it was decided to attempt the sinking of vertical shafts through the dump to obtain the desired samples. The advantages of this method of dump sampling are evident. Each shaft traverses numerous inclined layers of which the dump is composed and thus represents the accumulations of several years and not recent additions only. By dividing the excavated material into a number of samples, each representing five or ten feet of depth, the distribution of values from the top to the bottom of the dump may be studied, and each sample given its due weight in the final average, in proportion to the volume of ore represented. Finally, if the shafts be pushed to the bottom, not only are the deeply buried portions of the dumps reached and their value determined; but points in the original surface of the ground, on which the dump has been built, are located; from which points the contours of such buried surface may be determined, and the actual cross sections and volume of the dump computed with a greater degree of accuracy.

In order to limit the amount of material from each shaft and avoid the expense and delay of crushing unnecessarily large samples it was decided to sink shafts of as small diameter as practicable. In the loose and dry material of the dumps not only was a shaft lining required, but in all cases it proved necessary to drive the lining down every few inches, as the sinking progressed, or in other words some form of spiling or poling was essential to success. Suitable plank and timber for such work are not easily obtained in Bolivia and are expensive. Plank and timber also take up

much room in the shaft, requiring that the excavation be made larger than desirable. Under these conditions it was decided to try the experiment of using light iron rings instead of timber, backed by sheets of corrugated iron instead of a wooden lining. Corrugated iron is largely used in South America for building purposes and is therefore kept in stock and was readily obtainable in sufficient quantity and at reasonable cost. This material, cut into pieces of suitable size, made excellent poling boards, light and stiff and occupying but little space in the shaft. By reason of its smooth surface, the iron sheets drive easily with but little friction, requiring light blows only of a three-pound hammer. After experimenting with sheets of different sizes it was found that the best results were obtained with pieces about five feet long and a foot or less in width. To support this corrugated iron lining, elliptical rings of ⅜ by 1½ inch bar iron were used, the ends being welded together. The rings were made of elliptical rather than circular form to permit the large bottom rings of each set to be passed through the smaller upper rings. Also the elliptical form, for equal areas, gave the workmen more room for pick work and shovelling. These rings were supported by hooks of ⅜ inch rod as shown in the sketch, Fig. 3. After experiment with rings of different sizes, the dimensions indicated in the figure were adopted, the middle ring being 32 by 38 inches and the upper and lower ring respectively two inches smaller and larger each way. A shaft of these dimensions can be sunk rapidly and gives a sample of manageable size. With small men it proved practicable to use rings of less dimension than this. In one case a shaft was sunk successfully to a depth of 22 feet with rings averaging 22 × 28 inches. In all, 9 shafts were sunk in 8 weeks' time, the deepest being about 60 feet. This system of shaft lining, though composed of light material, possesses considerable strength and stiffness. The overlapping of the corrugated sheets and the double rings, tightly wedged together where the sections join, add much to the strength of the lining. The hooks are elements of weakness as these will straighten out at about one fifth of the ultimate breaking strength of the rods. The strength of these hooks may be increased by always inserting the point of the upper hook in the space between the ring and the corrugated iron sheets, and by fastening the point of the lower hook to the rod with a slip ring. The work of shaft sinking will be facilitated by having on hand extra hooks, shorter and longer than the standard length. It will be well also

to have a number of hooks with turn-buckles, to replace hooks that may be broken elsewhere than at the bottom of the shaft.

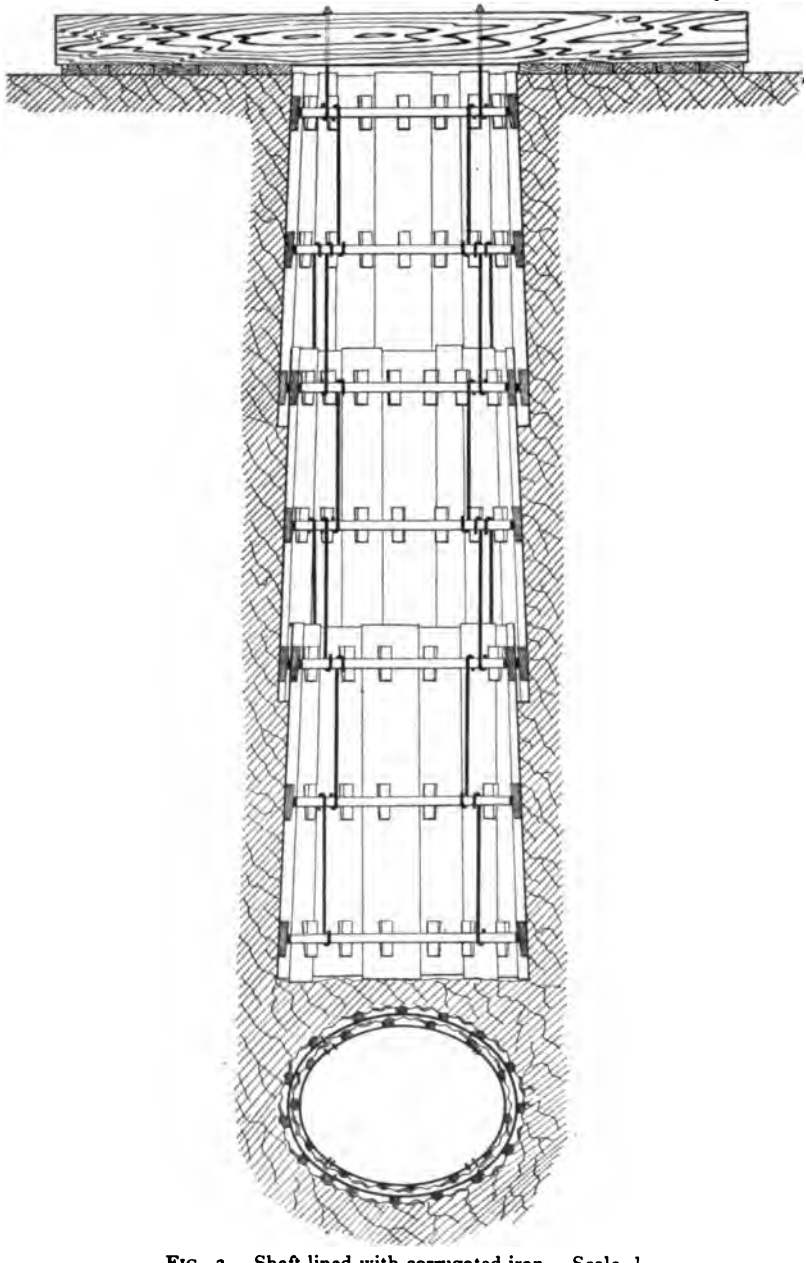


FIG. 3. Shaft lined with corrugated iron. Scale $\frac{1}{24}$.

SEGREGATION IN STEEL INGOTS.

By HENRY M. HOWE.

The solidification or freezing of every liquid or molten body, excepting pure substances like distilled water, pure unalloyed metals, etc., is a process of differentiation and concentration. Each layer as it starts to freeze spontaneously splits into two sublayers: (1) a more infusible and in general purer one, which freezes and thus attaches itself to the already frozen solid part, and (2) a more fusible and in general more impure one, which, staying molten, continues to form part of the remaining molten or liquid mother-metal or mother-liquor. The impurities, which have thus been concentrated into this second sublayer, immediately begin diffusing centerwards into the molten mother-metal; and the convection currents, due to the difference in density between the impure littoral sublayer which has thus been left along the cliff face or shore of the frozen continent, and the rest of the molten mass, aid this centripetal movement by sweeping this littoral layer either vertically upwards or vertically downwards along the shore. The result of this is that, on reaching the top or bottom as the case may be, the convection current sweeps axiswards along the top or bottom. As the currents from the several sides converge, say at the top, their convergence naturally leads to a downward movement along the axis. Thus are the impurities, which the differentiation of freezing concentrates in the littoral layer, swept by the convection currents into the axial region. This convection is often aided very greatly by the gas bubbles which are liberated in the act of freezing, because of the sudden fall of solvent power in passing from the molten to the solid state. These gas bubbles powerfully sweep the littoral layer upwards.

This process of differentiation and concentration is known as "segregation" or the flocking apart of the impurities. In case of ingots of steel the impurities which segregate most markedly are carbon, phosphorus, and sulphur, and the effect of their segregation is very serious. It is serious in case of carbon, because it is the carbon-content of the steel which is of the greatest importance in determining its strength, ductility, and other useful properties.

Segregation implies a change in the carbon-content, and a change of unknown degree; it further implies heterogeneousness of carbon-content, and hence of strength and ductility, and both experience and reason show that this heterogeneousness is in general very harmful, bringing the effective strength and effective ductility below those of the weakest and of the least ductile part respectively.

Segregation is serious in case of phosphorus and sulphur, because each of these elements is extremely harmful, and their concentration in a given region may there make the metal so bad that the value of the piece as a whole which contains this region may be greatly lessened or even destroyed.

It is, therefore, of importance to know the laws which govern segregation, and especially the influence of such variables as the rate of cooling and the size of the ingots on the degree of segregation.

To this end I have studied the results, many of them unpublished, of the examination of 168 steel ingots by many previous investigators, representing about 2,600 chemical analyses, most of which represent determinations of several different elements. Beyond this, I have had 8 cast-iron ingots and 17 steel ones made specially for this investigation, and I have studied their segregation. The total weight of these ingots which I have had made is about 22,000 pounds.

My results will be given in the Year Book of the Carnegie Institution of Washington, to which I am indebted for a grant for making this investigation, and probably in the *Transactions of the American Institute of Mining Engineers;* and a preliminary account of them is given in the *Engineering and Mining Journal* for November 30, 1907. They may be summed up as follows:

Ingot-size within the limits of 17 inches square and 2½ inches square has little constant influence on the degree of segregation. Ingots as much as 30 inches square, on the other hand, in many cases show great segregation. Even in ½ inch ingots the microscope shows great segregation.

In most of my experiments there was more segregation in the quickly cooled ingots than in those cooled slowly.

The region most enriched by segregation lies around the axis of the ingot, and in general at a distance from the ingot top of not less than 6 per cent. nor more than 28 per cent. of the total length of the ingot.

The region most impoverished by segregation is rarely if ever axial.

The enrichment in phosphorus and sulphur seems to be parallel with that of carbon, so that the isophoses and isotheis (lines of equal phosphorus and equal sulphur) are roughly parallel with the isocarbs.

On a general average, the greatest enrichment of phosphorus in any given ingot is about twice, and that of sulphur about thrice that of carbon; but these ratios vary greatly from case to case.

Unusual freedom from phosphorus or from sulphur does not seem to lessen the segregation of carbon, but if anything rather to increase it.

We should naturally expect that large ingot-size and slow cooling would favor segregation. The centerward migration covers a considerable distance, and the longer the time available for this migration, the greater should the migration be. What is the influence which prevents this natural effect? The explanation which I offer is at least simple. It is that large ingot-size, or rather the slow cooling to which that large size leads, by lessening the convection currents, directly restrains the vertical travel of the littoral layer. This vertical travel thus impeded is one very important part of the mechanism of segregation. Slow cooling further opposes segregation by favoring what I term the "landlocking" type of freezing as distinguished from the "onion" type. In the former, freezing proceeds by the out-shooting of pine-tree crystals, the tangled branches of which lock up much of the impure metal of the littoral layer, and thus prevent it from diffusing centreward, or in short from segregating. But in the onion type of freezing the successive deposition of smooth concentric layers does not thus obstruct the centripetal travel of the impurities.

Beyond this, the greater quiet may easily lead to surfusion, or to a greater degree of surfusion; and surfusion in turn should obstruct centripetal travel, because, once the overcooled mass does begin to freeze, its freezing naturally takes place not only rapidly but after the landlocking rather than the onion type.

MASONRY DAM FORMULAS.

By ORRIN L. BRODIE, C.E.

The method of analysis by which an economical cross-section of masonry dam can be most directly calculated was first devised by Mr. Ed. Wegmann,* and this method is set forth in the type-written notes prepared for students of civil engineering at Columbia University prior to 1905.

Objection has been raised to the "Wegmann Profile," or cross-section, in that the series of equations determining it lack the condition of uplift, due to water's penetrating the mass of the masonry, and of ice pressure exerted at the surface of the water in the reservoir, horizontally against the upstream face of the dam. Some difference of opinion may exist as to the relative importance of these last considerations, but when a structure of great responsibility is projected, present practice requires conservatism in design.

In general, it may be said in this respect, that the values assumed for the conditions imposed upon a design must be such that they are commensurate with the responsibility of the projected structure. A dam built above a populous district which, by its failure, would cause destruction of life and property is evidently of greater responsibility than a dam far removed from such a community. On the other hand, where a dam forms the main feature in an important system, although without the zone of potential catastrophe, it may, by failure, bring about widespread suffering and distress.

In the studies made for the Board of Water Supply of the City of New York, the procedure of arriving at the cross-section of the masonry portion of the Olive Bridge Dam, Ashokan Reservoir, and the cross-section of the proposed Kensico Dam involved, in the main, two steps:

First, the direct calculations fixing the most economical cross-section conforming to the given conditions and,

Second, comparative studies of cross-sections ranging between

*Cf. Report of F. Fteley to Chief Engineer B. S. Church of the Aqueduct Commissioners of the City of New York, on proposed Quaker Bridge Dam, dated July 25, 1887; also Mr. Wegmann's, "Design and Construction of Masonry Dams."

this minimum cross-section and one like that of an existing masonry dam of equal responsibility. The Wachusett Dam was chosen for comparison, as its cross-section represents a typical conservative design of recent date.

The purpose of this article is to present series of formulas deduced by the writer at different times, in connection with the above-mentioned work, as occasion demanded. They contain various imposed conditions and may be divided into two kinds, viz., formulas for design and formulas for investigation of a given design or cross-section. The formulas and conditions were used in part in these different studies made for the Board of Water Supply.

(The Board of Water Supply consists of Commissioners J. Edward Simmons, Charles N. Chadwick and Charles A. Shaw; Mr. J. Waldo Smith is Chief Engineer and Mr. Alfred D. Flinn is Department Engineer of Headquarters Department. The Designing Division is in charge of three Designing Engineers, who report to the Headquarters Department Engineer. Mr. C. E. Gregory, Designing Engineer, directs the principal studies and investigations relating to the reservoir work.)

It is believed that the arrangement and form of these formulas are most convenient for use, in fact they so proved where used in the above-mentioned investigations.

The nomenclature, in part, and the general form of the "formulas for investigation" appearing at the latter part of this paper, were suggested by notes on the same subject by Mr. Alfred D. Flinn.

As the "formulas for design" were developed from the type-written notes referred to in the first paragraph, these notes, essentially followed, will be rewritten and revised where pertinent.

This presentation will be divided into two sections, viz., Section I, an exposition of the general method of procedure in deriving the formulas, the simplest case being treated; and Section II, giving, in order, the various series of formulas for design, etc.

Section I. Method of Analysis.

(*a*) *Distribution of Pressure in a Rigid, Unyielding Joint.*

Fig. 1 exhibits a joint over which the pressure is uniformly distributed with the constant intensity, p. If l is the length of the joint, the total pressure exerted for each unit length of structure is pl and the center of pressure is at the center of the joint.

The treatment of the other cases will be expedited by the use of the following notation:

W = total pressure on base, or joint, l.

p = greatest intensity of pressure.

p' = least intensity of pressure, corresponding.

l = width of base, or joint length.

u = distance from edge of greatest intensity to the center of pressure.

In this discussion the material is assumed to be rigid but in reality it is to a certain degree elastic. This elasticity gives the distribution of stress an unknown and indeterminate law. Indeed

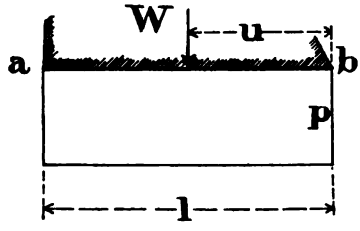


FIG. 1.

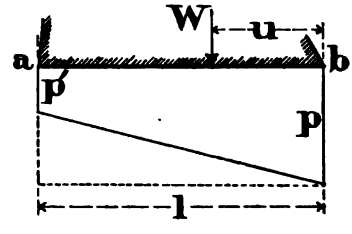


FIG. 2.

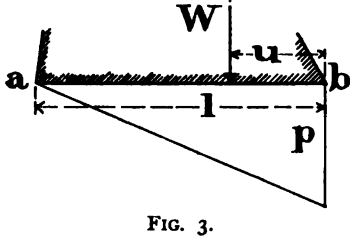


FIG. 3.

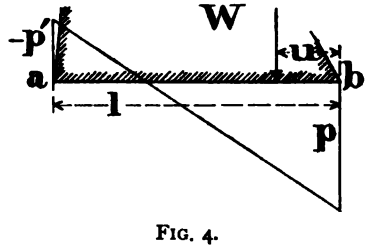


FIG. 4.

neither the direction nor intensity is actually known at any point. It is certain that the intensity must be zero at the edges of the joint, although it may increase with great rapidity to higher values very near the limits of the joint. Experiments of different kinds in recent years have been attempted to obtain more exact knowledge of this distribution of stress, but so far seem to have been unsatisfactory. Perhaps they have been undertaken upon too small a scale. However, they would seem to bear out the preceding observations. Inasmuch as the exact law of variation cannot be known, a law of uniform variation of normal stress is assumed in all practical treatments of masonry joints.

Fig. 2 exhibits the general case of compression throughout the entire joint. The area of the trapezoid which represents the total pressure in the joint is:—

$$\frac{(p+p')}{2}l. \tag{1}$$

The static moment of the rectangle pl about a point $\frac{1}{3}l$ from p' is precisely the same as the static moment of the trapezoid about the same point, because the moment of the triangle $(p-p')l/2$ about that point is zero.

Hence:

$$W(\tfrac{2}{3}l - u) = \frac{pl^2}{6}, \quad \text{hence} \quad p = \frac{2W}{l}\left(2 - \frac{3u}{l}\right). \tag{2}$$

When $p' = 0$, as in Fig. 3, $u = \frac{1}{3}l$ and

$$p = \frac{2W}{l}. \tag{3}$$

Fig. 4 represents the case of tension over a portion of the joint in which the tension is shown by the triangle whose side is $-p'$.

Although the best hydraulic cement mortar has considerable tensile strength, the latter, together with the continued adhesion of the mortar to the aggregate in concrete, when used, is of uncertain value in this connection. The tensile strength is always neglected in considering the stability of masonry dams or other similar structures. This is an omission which is the more justifiable because it leads to an error on the side of safety.

In Fig. 4, therefore, the triangle whose base is $3u$ and altitude p is alone considered. By taking moments about b,

$$Wu = \frac{3pu^2}{2}, \quad \text{hence} \quad p = \frac{2W}{3u}. \tag{4}$$

It has been virtually assumed that the pressure on any joint is normal in direction, but it is probable that this can be considered as reasonably accurate only for horizontal joints in vertical, rectangular walls not subjected to lateral pressure. The lateral, usually horizontal, thrust of water or earth will induce a horizontal component in the resultant pressure on any joint.

Again, it has been believed by Rankine and others that the direction of pressure at the edges of the joints must be nearly parallel to the exterior surfaces of the masonry. This assumption re-

ceives a reasonable, approximate confirmation from the fact that acute edges of stone do not crack off in inclined faces of dams.

Some eminent French authorities and others have therefore assumed that this case may be treated as shown in Fig. 5:

The horizontal joint is shown by *ac* and the inclined pressure by *R*. The obliquity of the latter is β. The triangular mass *cbd* is ignored and *ab* is taken as the effective area of the joint.

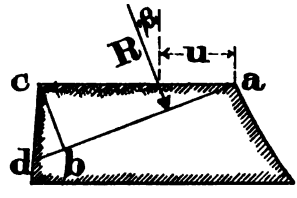


FIG. 5.

If $ac = l$, the joint over which *R* is distributed is $l \cos \beta$. The preceding formulas, equations (2), (3) and (4), are still applicable to this case by simply substituting:

$$l \cos \beta \text{ for } l; \quad u \cos \beta \text{ for } u; \text{ and } R \text{ for } W.$$

These equations will then become:

$$p_1 = \frac{2R}{l \cos \beta} \left(2 - \frac{3u}{l} \right), \tag{2_1}$$

$$p_1 = \frac{2R}{l \cos \beta}, \tag{3_1}$$

$$p_1 = \frac{2R}{3u \cos \beta}. \tag{4_1}$$

The same results are attained by supposing that the joint is serrated by planes alternately parallel and normal to *ab* and that the former portions of the serrations, only, receive the pressure *R*.

It is not at all clear on what grounds the assumptions, on which equations (2_1), (3_1) and (4_1) are based, can be supported. The pressure *R* is distributed over the joint *ac* and not over *ab*, nor are the positions of the serrations parallel to *ab*. It is manifestly improper, therefore, to make the unwarranted assumption even on the grounds of increased safety. It is true that the intensities in the vicinities of *a* and *c* are probably approximately parallel to the exterior surfaces at these points, but such probabilities tend to vitiate the assumptions in question.

If it is necessary, therefore, to assume that the pressures on a joint are all parallel, it is only reasonable and satisfactory to take the resultant pressure distributed over the actual joint (whether

horizontal or inclined), according to the law of uniformly varying intensity.

(*b*) *Development of Formulas for Design.*

Six series of formulas will be given and designated by the letters A, B, C, D, E and F, respectively. Each series conforms to a given set of conditions with respect to external forces. Series A will be developed only as the method of procedure remains, in general, the same for all series.

Series F forms a set of general equations for condition of water overtopping dam. These were used in part when investigating methods for prosecuting construction of Olive Bridge Dam.

The following nomenclature will apply to the formulas:

"Toe" or "Front" will signify down-stream face.

"Heel" or "Back" will signify up-stream face.

All linear distances are expressed in feet and areas in square feet.

In connection with the following see Figs. 9 to 13.

Nomenclature.

L = the width of the top of the dam cross-section.

l = length of a horizontal joint of masonry, *to be determined.*

l_0 = *known* length of the joint next above joint of length l.

h = depth of a course of masonry (vertical distance between l_0 and l).

P = line of pressure, reservoir full.

P' = line of pressure, reservoir empty.

u = distance from front edge of the joint l to the point of intersection of P with the joint l, measured parallel to joint l.

y = distance from back edge of the joint l to the point of intersection of P' with the joint l, measured parallel to joint l.

y_0 = distance from back edge of the joint l_0 to the point of intersection of P' with the joint l_0, measured parallel to joint l_0. (See Fig. 7.)

v = distance between P and P' at the joint l, measured parallel to joint l.

γ = weight in pounds of a cubic foot of water (62.5).

γ' = weight in pounds of a cubic foot of mud (75–90).

Δ = ratio of unit weight of masonry to unit weight of water (often assumed as 7/3).

$\Delta\gamma$ = weight in pounds of a cubic foot of masonry.

H = head of water on joint l (vertical distance of joint l below water surface).

H' = depth of earth back fill over joint l on front.

H_1 = head of water on joint l when ice acts at surface of water.

$H - H_1$ = rise of water level, due to flood, wave, etc., above normal level for full reservoir. (See Fig. 10.)

h_1 = head of water above mud level (liquid mud of weight γ').

h_2 = head of liquid mud on joint l, on back.

a = vertical distance from the top of the dam to the surface of water (flood).

a_1 = vertical distance from the top of the dam to the surface of water when ice is considered (a_1 generally exceeds a).

b = vertical distance from water surface to top of dam when dam is overtopped. (See Fig. 13.)

c = ratio of upward thrust intensity, due to hydrostic head H (or H_1, or $h_1 + h_2$), assumed to act at heel of joint l (usually assumed as 2/3).

$T\gamma$ = horizontal ice thrust at water surface in pounds (47,000).

(The value here given, for example, was used in studies for design cited. Our present lack of exact data in regard to ice pressures prevents more than a speculation from being made as to a definite value to be assigned in any case.)

$D\gamma$ = horizontal dynamic thrust of water in pounds.

$E\gamma$ = thrust of earth back fill in pounds (on front).

$W_v\gamma$ = vertical pressure on inclined upstream face above joint l, in pounds.

A_0 = total area of cross-section of dam above joint l_0.

A = total area of cross-section of dam above joint l.

t = batter of upstream face for vertical distance h.

s = distance of line of action of $W_s\gamma$ from upstream edge of joint l, measured parallel to joint l.

δ = angle that $E\gamma$ makes with horizontal. (See Fig. 12.)

α = angle of slope of downstream face of dam with horizontal. (See Fig. 12.)

β = angle R makes with the vertical.

p = maximum allowable pressure intensity at toe (in pounds per square foot).

q = maximum allowable pressure intensity at heel (in pounds per square foot) (p is assumed less than q) p and q may be used to signify the calculated, existent pressure intensities corresponding to P and P' respectively, for the joint l.

f = the coefficient of friction for masonry on masonry (usually 0.6 to 0.75.)

S = the shearing resistance of the masonry per square unit.

$F = \frac{\gamma H^2}{2}$ = the horizontal static thrust of the water in pounds.

$M = \frac{\gamma H^3}{6}$ = the moment of F about any point in the joint l.

$W = A\Delta\gamma$ = the total weight, in pounds, of masonry resting on the joint l.

$W_0 = A_0\Delta\gamma$ = the total weight, in pounds, of masonry resting on the joint l_0.

R = the resultant of F and W.

R' = the resultant of the reactions.

$\frac{cHl\gamma}{2}$ = upward thrust of water on base l.

In Figures 9 to 13, inclusive, hydrostatic pressures are indicated by triangular and trapezoidal areas included within dotted lines. Ice pressure is indicated in Figs. 10 and 11, to contrast H_1 with H.

As before, a unit length of one foot of dam will be considered. Then the letters T, D, E, W_s, A, A_0 and H^2 would signify volumes.

It will be observed that, where possible, the several equations will have been cleared of the term $\Delta\gamma$ (thereby simplifying actual calculations).

c in the above table, in a manner, may be considered to provide for an assumption of a certain proportion of the joint's area

being subjected to upward water pressure; and the distribution (as evidenced by $cHl\gamma/2$, varying from a maximum intensity at the heel to zero intensity at the toe) is assumed in view of the facts that the tendency to open the joint would begin at the heel and a zero intensity of upward pressure at the toe would presuppose an opening with consequent flow at that point. As the dam would then be failing in its chief function, i. e., to retain water, this flow is not considered to exceed a slight seepage.

There are four ways in which a masonry dam may fail:

1. By overturning;
2. By crushing;
3. By sliding or shearing;
4. By rupture from tension.

Another cause, viz., poor foundation, might be aptly mentioned here, although a feature to be guarded against by proper construction, as actual foundation conditions are learned only as construction progresses.

To insure ample safety against all causes of failure enumerated above we shall require the cross-sections which are to be determined to comply with the following conditions.

1. The lines of pressure must be within the center third of the cross-section whether the reservoir be full or empty.

2. The maxima pressures in the masonry or on the foundations must never exceed certain limits.

3. The friction between the dam and its foundation or between any two parts into which the dam may be divided by a horizontal plane, must be sufficient to prevent sliding.

The first of the above conditions precludes the possibility of tension and insures a factor of safety of at least two [if, by "factor of safety" is meant the ratio of resultant moment of vertical components to resultant moment of horizontal components of forces, moments taken about the downstream edge of the joint as an axis, or center of moments. This ratio differs from that of resisting moment to overturning moment except when uplift, referred to before, is ignored].

If the lines of action and magnitude of F and W are such that their resultant cuts the joint at e (Fig. 6), the limit of the middle third of l, the following equation of moments will hold:

$$\frac{FH}{3} = \frac{Wl}{3}. \qquad (5)$$

In this case, for the dam to be on the point of overturning, F must be doubled so that the center of pressure will be moved to b, the limit of the joint. $(u + v)/v =$ "factor of safety," so called. Strictly speaking the factor of safety, in this case, will be less than two because the material in the vicinity of b will crush considerably before rotation can take place about that point.

The ratio of resisting moment to overturning moment, about the toe, may be less than the factor of safety as above found, for in general, the increase of the resultant of the overturning forces necessary to cause the center of pressure to be moved to b, the limit of the joint, is the measure of the factor of safety against overturning, so that $(u + v)/v$ applies only to the factor of safety as first defined. The foregoing remarks are with special reference to effect of uplift on overturning forces.

The friction and the shearing resistance in any horizontal section or joint will resist any tendency to move the upper portion of the dam over the lower. The following relations must then hold if equilibrium is maintained:

$$F \lessgtr fW + Sl. \tag{6}$$

In general the left-hand member of equation (6) signifies the horizontal component of the resultant force.

Although S has an appreciable value for good masonry, and especially for monolithic masses of "cyclopean masonry," practically, the value is indeterminate. Numerous attempts have been made to write expressions for S, but the most rational depend upon the trapezoidal law of the distribution of normal stress and seem unsatisfactory from a practical standpoint. That value is therefore considered practically indeterminate and it is ordinarily ignored. This omission correspondingly increases frictional stability.

Equation (6) then takes this form:

$$F \lessgtr fW. \tag{7}$$

The value of f is usually taken, as before noted, at 0.6 to 0.75.

Figure 6 and Eq. (7) also give at the limit,

$$f = \frac{F}{W} = \tan \beta. \tag{8}$$

It will be found that the front faces of dams are much more in-

clined from the vertical line than the back faces. Hence, if the directions of intensity are approximately parallel to the faces near the limits of the joints, it follows that the intensity of pressure near the front face in any joint is more inclined and proportionally greater than that near the back face.

Since it is usual to ignore the horizontal or parallel component of pressure in any joint and treat only the vertical or normal component, it at once follows that the greatest vertical or normal working intensity of pressure must be less for the front face than for the back. In other words, we recognize considerably less than

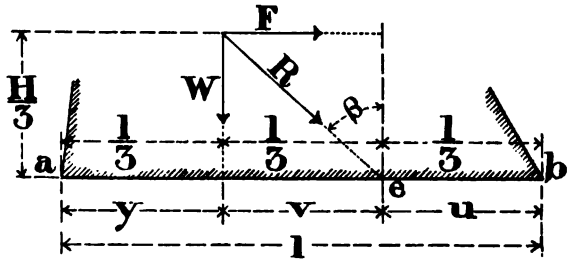


FIG. 6.

the resultant intensity at the front face and nearly the resultant intensity at the back face and make compensation for the error by using proportional values of working stresses.

Also, because the back or upstream face of the dam is only slightly inclined from a vertical line the weight of the volume of water resting on the back face is not considered and the resultant pressure of the water is consequently taken as horizontal. This omission is a small error on the side of safety, since the weight of water would produce a moment about the center of pressure opposite to the overturning moment of the horizontal water pressure.

If only the static pressure of the water be considered, for the time being, as determining the thickness, theory would require a zero thickness of masonry, only, at the water surface.

For various reasons the top of the dam must be carried a few feet above the water surface to give at least a convenient passage-way. For this purpose the width and super-elevation of the top of dam will be equal to $\frac{1}{10}$ the height of the dam, with a minimum of 5 feet for the former and maximum of 20 feet for the latter. It is evident that the values for these dimensions may be assumed for any given

case, being determined by local conditions, such as necessity for a highway over top of dam, fluctuations in reservoir level due to floods, etc., economic conditions, etc.

No equations expressing the three conditions of stability can be written; but the thickness must first be found by aid of the first two and an independent test must be made of the third.

It will usually be found that if the first two conditions are fulfilled, the results will meet the requirements of the third.

The cross-section will be determined by finding the required thickness, or width of base, at joints separated by any desired vertical distance. (It will usually be found convenient, for high dams, to assume $h = 10$ feet between rectangular top portion and a joint about 100 feet from top, h being thence increased to value of 20 feet or 30 feet.)

In order that the cross-section may enclose the least area, the limiting condition of stability must be expressed for each joint.

As shown by Figure (6) the total width of base (or length of joint) l, is composed of the three parts u, v and y.

Hence follows the fundamental equation,

$$l = u + v + y. \tag{9}$$

If M is equal to $FH/3$ (signifying the total overturning moment in a more general way), for stability, the following must obtain:

$$M = Wv.$$

Hence

$$v = \frac{M}{W}. \tag{10}$$

(Moments are taken about the point in the joint l, distant u from the front edge of the joint.)

The method of procedure to be followed will make the cross-section of polygonal outline, since it will be composed of trapezoids whose bases will be l and l_0 and heights h. Hence

$$W = W_0 + \left(\frac{l + l_0}{2}\right) h\Delta\gamma$$

or

$$A\Delta\gamma = A_0\Delta\gamma + \left(\frac{l + l_0}{2}\right) h\Delta\gamma$$

or

$$A = A_0 + \left(\frac{l + l_0}{2}\right) h \tag{11}$$

and

$$v = \frac{\frac{M}{\Delta\gamma}}{A_0 + \left(\frac{l + l_0}{2}\right) h} \tag{12}$$

and substituting v in Eq. (9) will give :

$$l = u + \frac{\frac{M}{\Delta\gamma}}{A_0 + \left(\frac{l + l_0}{2}\right) h} + y. \tag{13}$$

After the proper values of u and y are determined, Eq. (13) will enable the width of any base to be found.

In the upper portions of the dam the greatest pressures in any joint will be materially less than the allowed working stresses and hence it will be necessary to use the first condition of stability, so as to keep the center of pressure within the middle third of the joint, or base. Depths may then be found at which the intensities at the edges of the joints will become successively equal to p and q when the center of pressure in each case is at the limit of the middle third of the joint.

The value of u will cease to be $\frac{1}{3}\ l$ after the limit of pressure intensity p has been reached and similarly y will take the value of $\frac{1}{3}l$ for the last time when the corresponding intensity becomes equal to q.

Below each of the joints fixed by these conditions, the center of pressure will cease to approach as near as $\frac{1}{3}\ l$ to the respective face and its excursion induced by the alternate filling and emptying of the reservoir will be less than one third the width l of the base.

(It may lend to clearness to see Figures 9, 10 and 11 in connection with the following :)

In this analysis the first series, series A, will be deduced.

At and near the top of the dam, *i. e.*, at and above the joint which carries an amount of material in excess of that required by the water pressure the conditions will be:

$$u \geqq \tfrac{1}{3}l \quad \text{and} \quad y \geqq \tfrac{1}{3}l. \tag{14}$$

After the pressure limits p and q are reached the requisite values of u and y are to be taken from Eq. (2). These values are:

$$u = \frac{2l}{3} - \frac{pl^2}{6A\Delta\gamma}, \tag{15}$$

$$y = \frac{2l}{3} - \frac{ql^2}{6A\Delta\gamma}, \tag{16}$$

in which the expression for A is to be taken from Eq. (11).

For the reasons already given the upper part of the cross-section will show an excess of material and will be a rectangle in form down to a joint where one of the preceding limiting conditions will exist. Below that joint trapezoidal portions of the cross-section will be found.

At the base of the rectangular portion

$$l = l_0 = L, \quad u = \frac{L}{3}, \quad y = \frac{L}{2}.$$

Substituting in Eq. (13) and remembering that $A_0 = 0$

$$L = \frac{L}{3} + \frac{\frac{H^3}{6\Delta}}{L(H + a)} + \frac{L}{2}$$

or

$$H = \sqrt[3]{\Delta L^2(H + a)}. \tag{17}$$

If

$$H = h, \text{ then } a = 0 \quad \text{and} \quad h = L\sqrt{\Delta} \tag{17'}$$

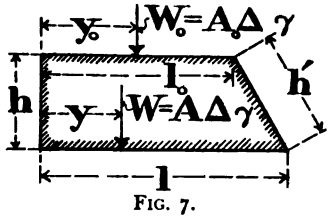


FIG. 7.

The front face h' (Fig. 7) must now be sloped, but the back face should be kept vertical until y becomes equal to $\frac{1}{3}l$. In order to find y, moments must be taken about the back face for A_0, and the trapezoid $(l + l_0)h/2$.

The latter moment is:

$$\frac{l_0^2 h}{2} + \frac{h}{2}(l - l_0)\left[l_0 + \frac{l - l_0}{3}\right] = \frac{h}{6}(l^2 + ll_0 + l_0^2)$$

$$\therefore Ay = A_0 y_0 + \frac{h}{6}(l^2 + ll_0 + l_0^2).$$

By introducing the value of A from Eq. (11) into the preceding, and reducing, there follows:

$$y = \frac{A_0 y_0 + \frac{h}{6}(l^2 + ll_0 + l_0^2)}{A_0 + \frac{l + l_0}{2}h}. \qquad (18)$$

By substituting this value for y and $u = \frac{1}{8}l$ in Eq. (13) and reducing,

$$l^2 + \left(\frac{4A_0}{h} + l_0\right)l = \frac{1}{h}\left(\frac{H^3}{\Delta} + 6A_0 y_0\right) + l_0^2. \qquad (19)$$

In Eq. (19) $y_0 = \frac{1}{2}L$ for the joint at the bottom of the rectangular portion. Eq. (19) must be used for all joints down to that one for which $y = \frac{1}{8}l$, below which the back face will cease to be vertical.

For a distance below this joint, $u = \frac{1}{8}l$ and $y = \frac{1}{8}l$ and both faces will slope. Hence, by introducing this value of u and y in Eq. (13),

$$l^2 + \left(\frac{2A_0}{h} + l_0\right)l = \frac{H^3}{\Delta h}. \qquad (20)$$

In all the preceding equations the greatest intensity of pressure has at no point reached the permissible limit; but the repeated application of Eq. (20) will finally determine a joint in which the greatest intensity will equal p, since p is less than q. It will be necessary, therefore, to determine the greatest intensity after each application of Eq. (20) and when it is found equal to p, the value of u must be taken from Eq. (15), while y remains equal to $\frac{1}{8}l$. By placing these values in Eq. (13) and by remembering that $A = A_0 + (l + l_0)h/2$,

$$l^2 = \frac{\gamma H^3}{p}. \qquad (21)$$

This equation is to be used until the limiting pressure q is reached at the back face of the dam, from which joint the values of u and y given by Eqs. (15) and (16) are to be inserted in Eq. (13).

This operation will give after reduction:

$$\left(\frac{p + q}{h\Delta\gamma} - 1\right)l^2 - \left(\frac{2A_0}{h} + l_0\right)l = \frac{H^3}{\Delta h}. \qquad (22)$$

Eq. (22) will determine the length of all remaining joints of the cross-section whatever may be its height.

These five main equations, then, viz., Eqs. (17), (19), (20), (21) and (22) will give the lengths of all joints from the top of the cross-section to the bottom. (In Eq. 17, joint length is usually assumed.)

The slope of the back face throughout that portion of the structure to which Eqs. (20), (21) and (22) apply, still remains to be found; it will have different rates at different points.

The methods of finding the first two equations, viz., Eqs. (17) and (19), involve the determination of y with respect to the vertical, back face [cf. Eq. (18)]; but, in the succeeding equations, that quantity has reference only to the back edge of the joint in which it is found. No condition is introduced by which it is located in reference to the back edge of the joint l_0. That location will now be made:

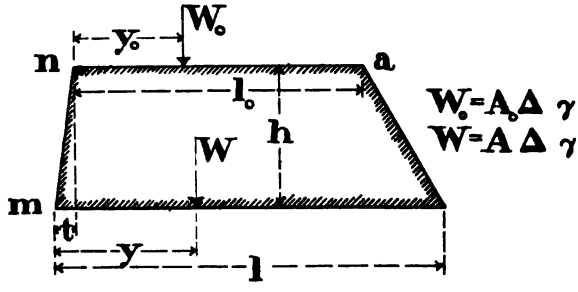


FIG. 8.

In Fig. 8, mn represents the back face of the dam and and t is the batter to be determined by taking static moments of A and A_0 about the back edge, m, of the joint.

The trapezoid of the figure is composed of the triangles $ht/2$ and $(l - l_0 - t)h/2$ and the rectangle hl_0.

By taking moments about the edge m,

$$Ay = A_0(y_0 + t) + \frac{ht^2}{3} + l_0 h\left(\frac{l_0}{2} + t\right) + \frac{h}{6}(l - l_0 - t)(l + 2l_0 + 2t). \quad (23)$$

For Eqs. (20) and (21) the value of y must be, as before, taken equal to $\frac{1}{3}l$, while A has the usual value of $A_0 + (l + l_0)h/2$. Substituting these in Eq. (23) and reducing:

$$t = \frac{2A_0(l - 3y_0) - hl_0^2}{6A_0 + h(2l_0 + l)}. \quad (24)$$

For the joints to which Eq. (22) applies the value of y is to be taken from Eq. (16) as was done before. In this case:

$$Ay = \tfrac{2}{3}l\left[A_0 + \left(\frac{l + l_0}{2}\right)h\right] - \frac{ql^2}{6\Delta\gamma}.$$

By substituting this value in the first member of Eq. (23) and reducing:

$$t = \frac{A_0(4l - 6y_0) + l^2\left(h - \frac{q}{\Delta\gamma}\right) + l_0h(l - l_0)}{6A_0 + h(2l_0 + l)}. \qquad (25)$$

After the value of l is found by the use of Eqs. (20), (21) or (22), t can at once be determined for the same joint by either Eq. (24) or Eq. (25).

In this manner an entire theoretical cross-section can be determined. It will be noticed that the location of the center of pressure in the middle third of the joint is the governing condition in the upper part of the dam, while the lower portion is fixed by the limiting pressures p and q. The difficulties preventing the forming of a simple working equation for the entire cross-section arise from the fact that the governing conditions are not introduced simultaneously nor in the same joint.

By taking h of the proper value, a polygonal cross-section may be determined by the preceding formulas. This cross-section can be then modified by drawing what may be called "mean" lines, straight, broken or curved along the theoretical faces so as to adapt the latter to a practical arrangement and treatment of the joints and facing blocks, which may be of cut stone or concrete.

The conditions which have governed the analysis are essentially those of Rankine, *i. e.*, the center of pressure has in all cases been kept within the middle third of the joint and the greatest intensity of pressure, either at the front face or back, has not been allowed to exceed the limit p or q.

SECTION II. FORMULAS.

Series A, B, C, D, E and F and Formulas for Investigation.

As noted earlier in this paper, the above designations refer to the six separate series of formulas. These series are here set forth in suitable form for easy reference and use. As they have been developed by the method just outlined it is considered unnecessary

to follow out the derivation of each series, although there exist some detailed differences in the treatment of the various series. These details, however, would become evident to anyone following the deductions throughout.

Various conditions of "loading," with approved assumptions, such as pressure due to expanding ice at the water surface, upward water pressure on the base, etc. (referred to in the table of nomenclature previously given) have been introduced and are specifically stated for each case.

It will be recalled that Eq. (13) in Section I is the fundamental expression for finding the length, l, of any joint; and, as the several conditions are introduced, that the "M" in Eq. (13) must in each case signify the total overturning moment and not merely the moment of the static water pressure on the back.

The development of a cross-section, by any one of the following series, may comprise five stages, each stage representing the introduction of a governing condition. Hence, for each stage there obtains a main equation for finding the length of joint, l, each main equation being supplemented by secondary equations for y, u, and t; p and q.

It may be necessary to employ more than one of the series of equations in determining a cross-section.

For ready reference, the five stages will be set forth and depicted in order as follows:

Stages in Development.

Stage I. — This stage, it will be remembered, extends from the top of the dam to the joint where the front face commences to batter. It is the rectangular section. $y > \frac{1}{8}L$; $u \geqq \frac{1}{8}L$ (see Fig. 9). (Ice pressure is purposely omitted in Fig. 9 to prevent confusion of letters in small space.)

Stage II. — This stage extends from the lower limit of Stage I. to the point where the back face commences to batter. $u = \frac{1}{8}l$; $y \geqq \frac{1}{8}l$ (see Fig. 10).

Stage III. — This stage extends from the lower limit of Stage II. to the point where the intensity of pressure on the toe has reached the maximum allowable intensity. In this stage $u = \frac{1}{8}l$; $y = \frac{1}{8}l$ (see Fig. 11).

Stage IV. — This stage extends from the lower limit of Stage III. to the point where the pressure intensity on the heel has

reached the maximum allowable intensity. For this stage $u > \frac{1}{3}l$; $y = \frac{1}{3}l$. (See Fig. 11.)

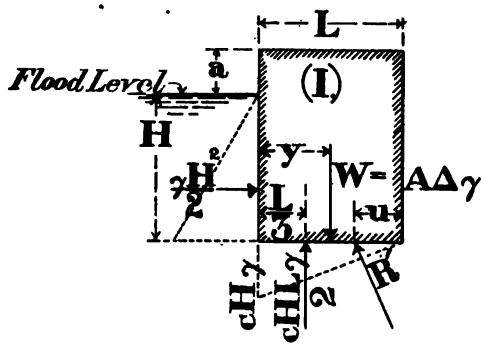


FIG. 9.

Stage V. — In this stage the limiting intensities of pressure at both toe and heel having been reached, $y > \frac{1}{3}l$; $u > \frac{1}{3}l$.

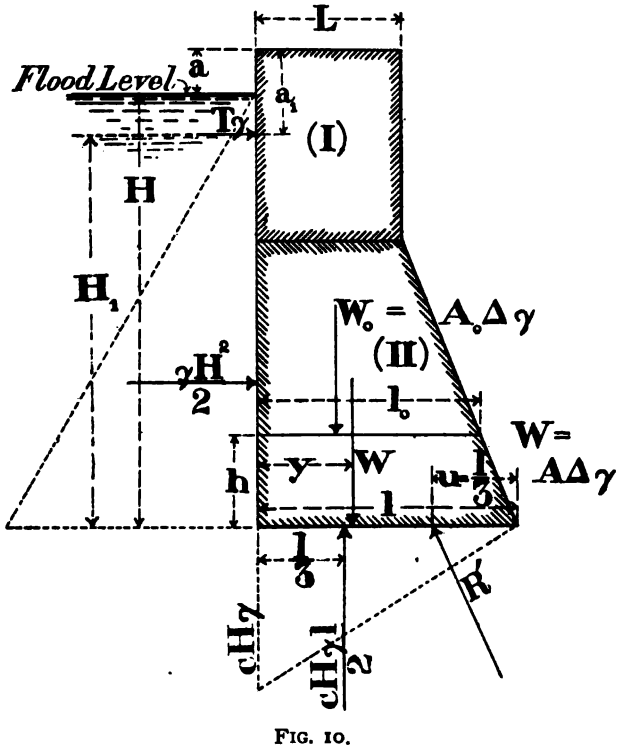


FIG. 10.

This stage extends from the lower limit of Stage IV. downward. (See Fig. 11.)

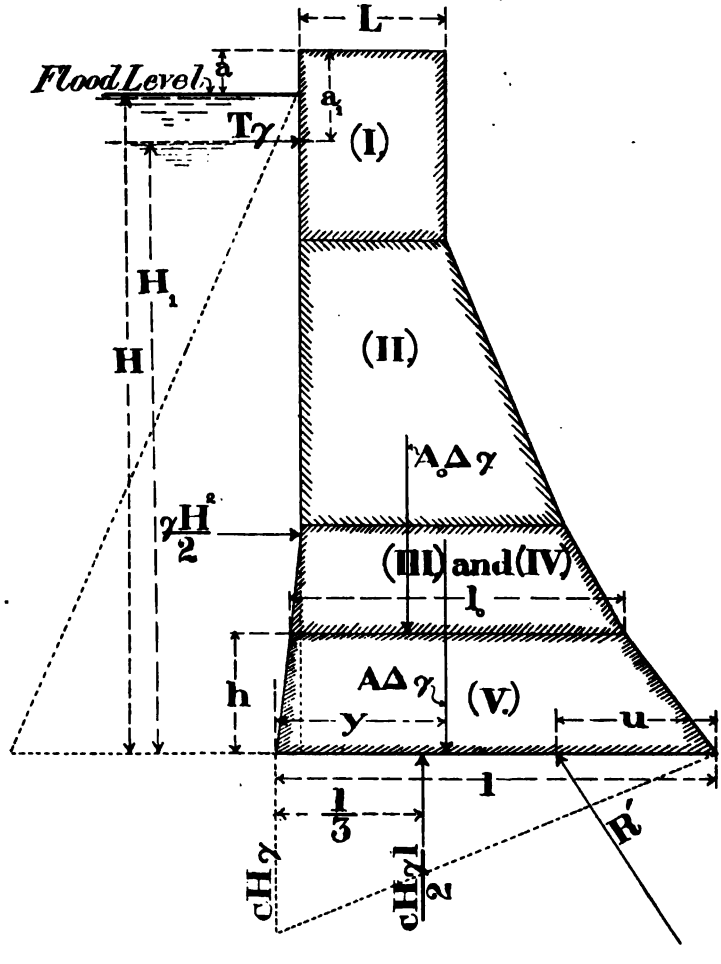


FIG. 11.

The following secondary formulas, supplementary to the main equations of all series, with substitutions as noted, are arranged in order corresponding with the preceding.

Equations Supplementary to all Series.

Stage I.

$u \geqq \frac{1}{3}L$

$y = \frac{1}{2}L$

$t = 0$

$$p = \frac{2\Delta\gamma A}{L}\left(2 - \frac{3u}{L}\right)$$

$$q = \frac{\Delta\gamma A}{L}$$

With condition of hydrostatic upward pressure on base obtaining, substitute the formulas in this column in place of those corresponding, as indicated.

$$p = \gamma\left(\frac{2\Delta A}{L} - cH\right)\left(2 - \frac{3u}{L}\right)$$

Stage II.

$u = \frac{1}{3}l$

$$y = \frac{A_0 y_0 + (l^2 + ll_0 + l_0^2)\frac{h}{6}}{A_0 + \left(\frac{l + l_0}{2}\right)h}$$

$t = 0$

$$p = \frac{2\Delta\gamma A}{l}$$

$$p = \gamma\left(\frac{2\Delta A}{l} - cH\right)$$

$$q = \frac{2\Delta\gamma A}{l}\left(2 - \frac{3y}{l}\right)$$

Stage III.

$u = \frac{1}{3}l$

$y = \frac{1}{3}l$

$$t = \frac{2A_0(l - 3y_0) - hl_0^2}{6A_0 + h(2l_0 + l)}$$

$$p = \frac{2\Delta\gamma A}{l}$$

$$p = \gamma\left(\frac{2\Delta A}{l} - cH\right)$$

$$q = \frac{2\Delta\gamma A}{l}$$

Stage IV.

$$u = \frac{2}{3}l - \frac{pl^2}{6\Delta\gamma A}$$

$$u = \frac{2}{3}l - \frac{pl^2}{3\gamma(2\Delta A - cHl)}$$

$y = \frac{1}{3}l$

Equations Supplementary to all Series — (continued).

With condition of hydrostatic upward pressure on base obtaining, substitute the formulas in this column in place of those corresponding, as indicated.

$$t = \frac{2A_0(l - 3y_0) - hl_0^2}{6A_0 + h(2l_0 + l)}$$

$$p = \frac{2\Delta\gamma A}{l}\left(2 - \frac{3u}{l}\right) \text{ (limiting intensity)} \qquad p = \gamma\left(\frac{2\Delta A}{l} - cH\right)\left(2 - \frac{3u}{l}\right)$$

$$q = \frac{2\Delta\gamma A}{l}$$

Stage V.

$$u = \tfrac{2}{3}l - \frac{pl^2}{6\Delta\gamma A} \qquad u = \tfrac{2}{3}l - \frac{pl^2}{3\gamma(2\Delta A - cHl)}$$

$$y = \tfrac{2}{3}l - \frac{ql^2}{6\Delta\gamma A}$$

$$t = \frac{A_0(4l - 6y_0) + \left(h - \frac{q}{\Delta\gamma}\right)l^2 + (l - l_0)hl_0}{6A_0 + h(2l_0 + l)}$$

$$p = \frac{3\Delta\gamma A}{l}\left(2 - \frac{3u}{l}\right) \text{ (limiting intensity)} \qquad p = \gamma\left(\frac{2\Delta A}{l} - cH\right)\left(2 - \frac{3u}{l}\right)$$

$$q = \frac{2\Delta\gamma A}{l}\left(2 - \frac{3y}{l}\right) \text{ (limiting intensity)}$$

If T enter following formulas, H above becomes H_1 (see Figs. 10 and 11).

The first column of formulas just given would apply, with condition of upward pressure on base due to hydrostatic head, if a proper value of u corresponding, be taken, that is if the excursion of force $A\Delta\gamma$, resulting from effect of all other forces on $A\Delta\gamma$ be considered, rather than the effect of all other forces on resultant vertical force.

It will be found expeditious to design a section, where ice pressure at level of full reservoir is to be considered in connection with water surface at some higher flood level, first by series of formulas containing T (cf. Series B and D) and then to investigate successive bases, or joints, thus obtained (beginning, for a high masonry dam, usually at a base, or joint, about 100 feet from the top of the dam) with series of formulas lacking T, or ice pressure

condition (cf. Series *A* and *C*). A base will ultimately be obtained by these supplementary "Flood level" calculations greater than the base at its same elevation and as previously determined by the "Ice Pressure" design.

Continuing with design by means of the "Flood Level" formulas to the bottom of maximum height required will determine the minimum cross-section area to meet conditions both of "Flood" and of "Ice." It should be remarked in this connection that when a reservoir level is rising due to flood conditions prevailing, it is evident that ice formation cannot develop, or, in other words, the two conditions cannot be co-existent, hence the difference in designation of hydrostatic heads corresponding. (See Figs. 10 and 11.)

Series A.

Conditions: Overturning moment due to *horizontal static water pressure on back of dam only.*

Stage I.

$$H = \sqrt[3]{\Delta L^2(H+a)}.$$

Stage II.

$$l^2 + \left(\frac{4A_0}{h} + l_0\right) l = \frac{1}{h}\left(\frac{H^3}{\Delta} + 6A_0 y_0\right) + l_0^2.$$

Stage III.

$$l^2 + \left(\frac{2A_0}{h} + l_0\right) l = \frac{H^3}{\Delta h}.$$

Stage IV.

$$l^2 = \frac{\gamma H^3}{p}.$$

Stage V.

$$\left(\frac{p+q}{h\Delta\gamma} - 1\right) l^2 - \left(\frac{2A_0}{h} + l_0\right) l = \frac{H^3}{\Delta h}$$

Series B.

Conditions: Overturning moment due to:

(*a*) *Horizontal static water pressure on back* and

(*b*) *Ice pressure applied at distance* (a_1) *from top.*

Stage I.

$$H_1 = \sqrt[3]{\Delta(H_1 + a_1)L^2 - 6TH_1}.$$

Stage II.

$$l^2 + \left(\frac{4A_0}{h} + l_0\right) l = \frac{1}{h}\left(\frac{H_1^3 + 6TH_1}{\Delta} + 6A_0 y_0\right) + l_0^2.$$

Stage III.

$$l^2 + \left(\frac{2A_0}{h} + l_0\right) l = \frac{1}{\Delta h}\left(H_1^3 + 6TH_1\right).$$

Stage IV.

$$l^2 = (H_1^3 + 6TH_1)\frac{\gamma}{p}.$$

Stage V.

$$\left(\frac{p+q}{h\Delta\gamma} - 1\right) l^2 - \left(\frac{2A_0}{h} + l_0\right) l = \frac{1}{\Delta h}(H_1^3 + 6TH_1).$$

Series C.

Conditions: Overturning moment due to:

(*a*) *Horizontal statie water pressure on back.*

(*b*) *Upward water pressure on base.* Pressure intensity decreasing uniformly from $cH\gamma$ at heel to zero intensity at toe.

Stage I.

$$H = \sqrt[3]{L^2[\Delta(H + a) - cH]}.$$

Stage II.

$$\left(1 - \frac{cH}{\Delta h}\right) l^2 + \left(\frac{4A_0}{h} + l_0\right) l = \frac{1}{h}\left(\frac{H^3}{\Delta} + 6A_0 y_0\right) + l_0^2.$$

Stage III.

$$\left(1 - \frac{cH}{\Delta h}\right) l^2 + \left(\frac{2A_0}{h} + l_0\right) l = \frac{H^3}{\Delta h}.$$

Stage IV.

(*a*) $$\left(1 - \frac{cH}{\Delta h}\right) l^3 + \left(\frac{2A_0}{h} + l_0\right) l^2 - \frac{\gamma H^3}{p}\left(1 - \frac{cH}{\Delta h}\right) l = \frac{\gamma H^3}{p}\left(\frac{2A_0}{h} + l_0\right),$$ which reduces to

(*b*) $$l^2 = \frac{\gamma H^3}{p}.$$

Stage V.

(*a*) $$\left(1 - \frac{cH}{\Delta h}\right)\left(1 - \frac{p+q}{h\Delta\gamma}\right) l^3 + \left(\frac{2A_0}{h} + l_0\right)\left(2 - \frac{p+q}{h\Delta\gamma} - \frac{cH}{h\Delta}\right) l^2 + \left[\frac{2A_0}{h}\left(\frac{2A_0}{h} + 2l_0\right) + \frac{H^3}{h\Delta}\left(1 - \frac{cH}{h\Delta}\right) + l_0^2\right] l$$

$$= -\frac{H^3}{h\Delta}\left(\frac{2A_0}{h} + l_0\right), \text{ which reduces to}$$

$$(b) \quad \left(\frac{p+q}{h\Delta\gamma} - 1\right) l^2 - \left(\frac{2A_0}{h} + l_0\right) l = \frac{H^3}{\Delta h}.$$

Series D.

Conditions: Overturning moment due to:

(*a*) *Horizontal static water pressure on back* (head = H_1).

(*b*) *Ice pressure applied at distance* (a_1) *from top.*

(*c*) *Upward water pressure on base.* Pressure decreasing uniformly from $cH_1\gamma$ at heel to zero intensity at toe.

Stage I.

$$H_1 = \sqrt[3]{L^2[(H_1 + a_1)\Delta - cH_1] - 6TH_1}.$$

Stage II.

$$\left(1 - \frac{cH_1}{\Delta h}\right) l^2 + \left(\frac{4A_0}{h} + l_0\right) l = \frac{1}{h}\left(\frac{H_1^3 + 6TH_1}{\Delta} + 6A_0 y_0\right) + l_0^2.$$

Stage III.

$$\left(1 - \frac{cH_1}{\Delta h}\right) l^2 + \left(\frac{2A_0}{h} + l_0\right) l = \frac{1}{\Delta h}(H_1^3 + 6TH_1).$$

Stage IV.

$$(a) \quad \left(1 - \frac{cH_1}{\Delta h}\right) l^3 + \left(\frac{2A_0}{h} + l_0\right) l^2 - \frac{\gamma(H_1^3 + 6TH_1)}{p}\left(1 - \frac{cH_1}{\Delta h}\right) l$$

$$= \frac{\gamma(H_1^3 + 6TH_1)}{p}\left(\frac{2A_0}{h} + l_0\right), \text{ which reduces to}$$

$$(b) \quad l^2 = \frac{\gamma(H_1^3 + 6TH_1)}{p}.$$

Stage V.

$$(a) \quad \left(1 - \frac{cH_1}{\Delta h}\right)\left(1 - \frac{p+q}{h\Delta\gamma}\right) l^3 + \left(\frac{2A_0}{h} + l_0\right)\left(2 - \frac{p+q}{h\Delta\gamma} - \frac{cH_1}{h\Delta}\right) l^2$$

$$+ \left[\frac{2A_0}{h}\left(\frac{2A_0}{h} + 2l_0\right) + \frac{(H_1^3 + 6TH_1)}{h\Delta}\left(1 - \frac{cH_1}{h\Delta}\right) + l_0^2\right] l$$

$$= -\frac{(H_1^3 + 6TH_1)}{\Delta h}\left(\frac{2A_0}{h} + l_0\right), \text{ which reduces to}$$

$$(b) \quad \left(\frac{p+q}{h\Delta\gamma} - 1\right) l^2 - \left(\frac{2A_0}{h} + l_0\right) l = \frac{H_1^3 + 6TH_1}{\Delta h}.$$

In the preceding series of equations it will be observed that the final expressions for l in stages IV and V, are very similar, and that the quantity c in equations (a) of these stages disappears in equations (b). Equations (b) of course, are to be used for purposes of calculation of cross-sections.

Series E.

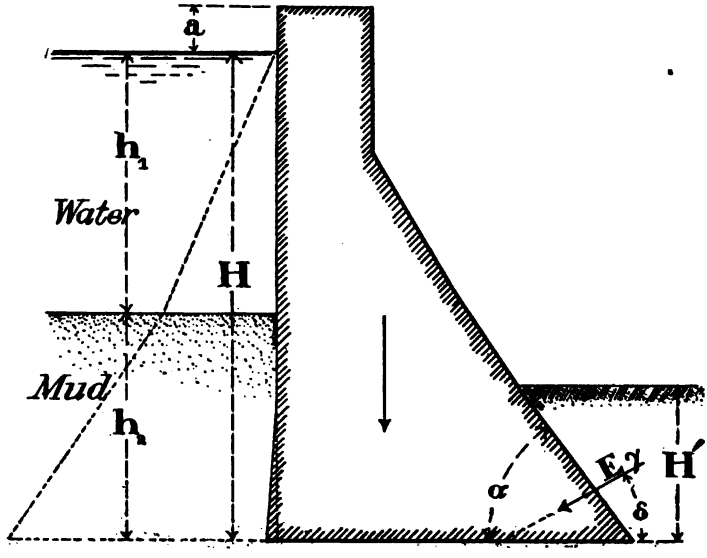


FIG. 12.

Conditions: (See Fig. 12) Ice pressure neglected in Fig. 12. Overturning moment due to:

(a) *Horizontal static water pressure on back* (head = h_1).

(b) *Ice pressure applied at distance* (a_1) *from top.*

(c) *Upward water pressure on base;* pressure intensity decreasing uniformly from $c(h_1 + h_2)\gamma$ at heel to zero intensity at toe.

(d) *Mud* (*liquid*) *pressure on back* (head h_2), commencing at distance h_2 above joint in question. Weight of mud = γ'. As before, if T be equated to zero, a_1 becomes equal to a, in formulas.

Stage I. — (h_1, of known value, h_2 to be determined.)

$$\frac{h_2^3\gamma'}{\gamma} + (h_1 + h_2)[3h_1h_2 + 6T + L^2(c - \Delta)] = L^2a_1\Delta - h_1^3.$$

Stage II.

$$\left[1-\frac{c(h_1+h_2)}{\Delta h}\right]l^2+\left(\frac{4A_0}{h}+l_0\right)l$$

$$=\frac{1}{\Delta h}\left[(h_1+h_2)(3h_1h_2+6T)+h_1^3+\frac{h_2^3\gamma'}{\gamma}+6A_0y_0\Delta\right]+l_0^2.$$

For trapezoidal section at top, make $A_0=0$ and $y_0=0$ and $l_0=L$ in Stage II. This applies generally.

Stage III.

$$\left[1-\frac{c(h_1+h_2)}{\Delta h}\right]l^2+\left(\frac{2A_0}{h}+l_0\right)l$$

$$=\frac{1}{\Delta h}\left[(h_1+h_2)(3h_1h_2+6T)+h_1^3+\frac{h_2^3\gamma'}{\gamma}\right].$$

Stage IV.

$$l^2=\frac{\gamma}{p}\left[(h_1+h_2)(3h_1h_2+6T)+h_1^3+\frac{h_2^3\gamma'}{\gamma}\right].$$

Stage V.

$$\left(\frac{p+q}{h\Delta\gamma}-1\right)l^2-\left(\frac{2A_0}{h}+l_0\right)l$$

$$=\frac{1}{\Delta h}\left[(h_1+h_2)(3h_1h_2+6T)+h_1^3+\frac{h_2^3\gamma'}{\gamma}\right].$$

From a study of the formulas thus far developed it will be observed that by reducing certain conditions to zero, with their corresponding quantities, the main equations of a given series reduce to those of a simpler series.

For instance —

In Series *B* make *T* (for ice pressure condition of loading) equal to zero and $H_1=H$ and $a_1=a$ and the main equations of that series reduce to Series *A* equations.

In Series *C*, by making *c* (for upward water pressure condition) equal to zero in main equations of Stages I, II, and III and also in equations (*a*) of Stages IV and V, the equations of Series *C* reduce to those of Series *A*.

Likewise, by making the proper eliminations and substitutions, Series *E* will reduce to Series *D*, *C*, *B* or *A*.

Series F.

This series consists of general formulas for a number of imposed conditions of loading. For any given case, the terms or factors expressing those conditions not appertaining must be eliminated by equating them to zero. (See Fig. 13.)

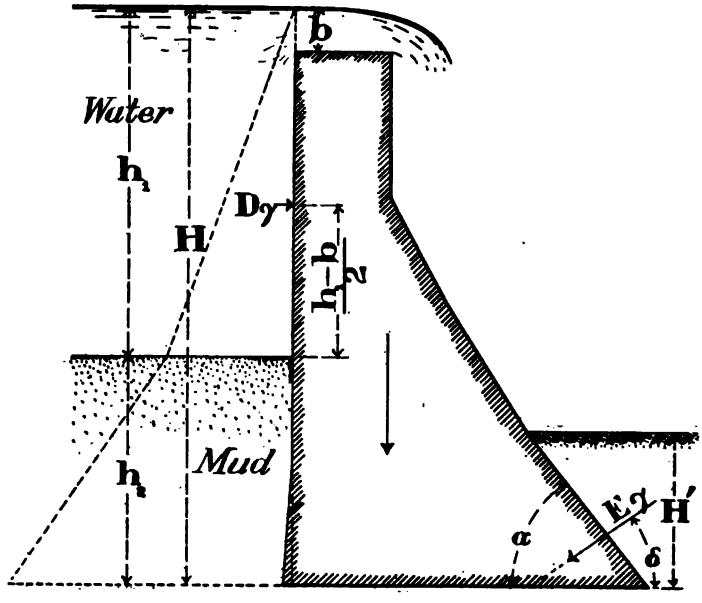


FIG. 13.

Conditions for General Formulas.

Overturning moment due to:

(*a*) *Horizontal static water pressure on back* (head = h_1).

(*b*) *Upward water pressure on base;* pressure intensity decreasing uniformly from $cH\gamma$ or $c(h_1 + h_2)\gamma$, at heel to zero intensity at toe.

(*c*) *Mud* (*liquid*) *pressure on back* (head h_2) as before.

(*d*) *Dynamic pressure of water*, $D\gamma$.

(*e*) *Water flowing over top of dam*, weight of water, of depth b, on top of dam being neglected.

For condition of water *not* overtopping dam, $b = 0$ and $D = 0$.

For condition of no dynamic pressure, $D = 0$.

For condition of no upward water pressure, $c = 0$.

For condition of no mud (i. e., mud being replaced by water) make $h_2 = 0$, $h_1 = H$.

Series F.

Stage I.

Rectangular cross-section at top or rectangular dam, $l = l_0 = L$. This may fall under either of two cases viz. —

Case (1)

Condition: $h_1 = H$; $h_2 = 0$.

$$H^3 + H[+ 3D - 3b^2 + L^2(c - \Delta)] = b(3D - 2b^2 - L^2\Delta).$$

Case (2)

Condition: h_1 of known value; h_2 to be determined.

$$\frac{h_2^3\gamma'}{\gamma} + 3Dh_2 + (h_1 + h_2)[3h_1h_2 + 3D - 3b^2 + L^2(c - \Delta)] = b(3D - 2b^2 - L^2\Delta) - h_1^3.$$

As in the preceding series, the value of H or h_2, of Stage I may be determined by several successive trial substitutions.

Stage II.

(*a*) Trapezoidal cross-section at top of dam or trapezoidal dam (spillway) front face battered. ($A_0 = 0$, $l_0 = L$ and $y_0 = 0$.) Note: For a triangular dam $l_0 = 0$, also.

$$\left[1 - \frac{c(h_1 + h_2)}{\Delta h}\right] l^2 + Ll = \frac{1}{\Delta h}\left[(h_1 + h_2)(3h_1h_2 - 3b^2) + h_1^3 + \frac{h_2^3\gamma'}{\gamma} + 2b^3 + 3D(h_1 + 2h_2 - b)\right] + L^2.$$

(*b*) Trapezoidal section continued (front face battered).

$$\left[1 - \frac{c(h_1 + h_2)}{\Delta h}\right] l^2 + \left(\frac{4A_0}{h} + l_0\right) l = \frac{1}{\Delta h}\left[(h_1 + h_2)(3h_1h_2 - 3b^2) + h_1^3 + \frac{h_2^3\gamma'}{\gamma} + 2b^3 + 3D(h_1 + 2h_2 - b) + 6A_0y_0\Delta\right] + l_0^2.$$

Stage III. — Both faces battered.

$$\left[1 - \frac{c(h_1 + h_2)}{\Delta h}\right] l^2 + \left(\frac{2A_0}{h} + l_0\right) l = \frac{1}{\Delta h}\left[(h_1 + h_2)(3h_1h_2 - 3b^2) + h_1^3 + \frac{h_2^3\gamma'}{\gamma} + 2b^3 + 3D(h_1 + 2h_2 - b)\right].$$

Stage IV. — Limiting intensity of pressure, p, introduced.

$$l^2 = \frac{\gamma}{p}\left[(h_1 + h_2)(3h_1h_2 - 3b^2) + h_1^3 + \frac{h_2^3\gamma'}{\gamma} + 2b^3 + 3D(h_1 + 2h_2 - b)\right].$$

Stage V. — Limiting intensities, p and q.

$$\left(\frac{p+q}{h\Delta\gamma} - 1\right)l^2 - \left(\frac{2A_0}{h} + l_0\right)l = \frac{1}{\Delta h}\left[(h_1 + h_2)(3h_1h_2 - 3b^2) + h_1^3 + \frac{h_2^3\gamma'}{\gamma} + 2b^3 + 3D(h_1 + 2h_2 - b)\right].$$

The increased number of overturning loads, then, tend to render the right hand members of the various equations more involved; though, after a little practice one may easily carry through a design with surprising rapidity. The slide rule may be used to great advantage and it is suggested that the results be tabulated as determined, in some such form as the following:

TABLE OF RESULTS.

$c =$, $T =$, etc.

Joint No.	H	H^3	h	$H+a$	A_0	A	l_0	l_0^2	l	y_0	y	u	t	p	q	A_0y_0	Etc.
1																	
2																	
3																	
4																	
Etc.																	

The effect upon the calculation of a cross-section of backfill on the downstream face could of course be cared for by introducing that condition into the preceding series of equations; but as this effect, as computed, would be, in any case, largely dependent upon assumptions which may vary widely and as the placing of backfill is generally a later consideration with respect to construction, the propriety of such introduction at that stage of design is questionable.

In the following formulas for *investigation*, therefore, the general conditions of an earth thrust acting at the downstream face and of a vertical component of thrust of material on the upstream, inclined face of the dam are introduced.

By any of these formulas, the position of the line of resistance

for any given cross-section and respective conditions may be determined with regard to any horizontal joint and its downstream edge; the value of u being the quantity to be sought.

Any condition may be disregarded by equating its term to zero.

The first expression below contains all of the conditions heretofore considered with the additional ones just stated; and from it follow the succeeding expressions for u. It should be remembered that the term T cannot be co-existent in any expression for stability with b and therefore with D. Nevertheless all of these terms are written with the understanding that the proper eliminations be always made. Three general group equations will be written.

Formulas for Investigation.

First, Conditions of retained mud, water, overtopping, etc. (see Fig. 13).

$$u = l - y - \frac{(h_1 + h_2)[3h_1h_2 + 6T - 3b^2 + cl(3y - l)] + h_1^3 + \frac{h_2^3\gamma'}{\gamma} + 2b^3 + 3D(h_1 + 2h_2 - b) - 6W_s(y - s) + 6E\left[(l - y)\sin\delta - \frac{H'}{3}\frac{\sin(\delta + \alpha)}{\sin\alpha}\right]}{6(W_s + E\sin\delta + A\Delta) - 3c(h_1 + h_2)l}.$$

Whence, for conditions of *retained mud, water, etc.*, but *no overtopping*, by making $b = 0$ and $D = 0$, there follows (see Fig. 12),

$$u = l - y - \frac{(h_1 + h_2)[3h_1h_2 + 6T + cl(3y - l)] + h_1^3 + h_2^3\frac{\gamma'}{\gamma} - 6W_s(y - s) + 6E\left[(l - y)\sin\delta - \frac{H'}{3}\frac{\sin(\delta + \alpha)}{\sin\alpha}\right]}{6(W_s + E\sin\delta + A\Delta) - 3c(h_1 + h_2)l}.$$

From this last expression for u, for conditions of *retained water, etc.*, but *neither mud nor overtopping*, by making $h_1 = H_1$; $h_2 = 0$, there is obtained :

$$u = l - y - \frac{6T + H_1^2 + cl(3y - l) - \frac{6W_s}{H_1}(y - s) + \frac{6E}{H_1}\left[(l - y)\sin\delta - \frac{H'}{3}\frac{\sin(\delta + \alpha)}{\sin\alpha}\right]}{6\left(\frac{W_s + E\sin\delta + A\Delta}{H_1}\right) - 3cl}.$$

As in equations for design, when $T = 0$, $H_1 = H$. (See Figs. 10 and 11.) If H' is of such depth that downstream batter of cross-section varies considerably, an approximate solution is possible by assuming some average batter for lower portion. The expression for earth thrust is general, as is seen. After u is determined for each joint, the intensities of maxima pressures can be determined for the given cross-section, the general expression for p, corresponding to above expressions for u, being:

$$p = \frac{2\gamma}{l}\left[W_s + E\sin\delta + A\Delta - \frac{c(h_1 + h_2)}{2}l\right]\left(2 - \frac{3u}{l}\right).$$

In connection with the computation for the value of y in an investigation, as indicated above, it is necessary to obtain the position of the centroid of a trapezoid with respect to the back, or upstream, edge of the joint in question. The following expression for x, in connection with Fig. 14, may prove convenient:

$$x = \frac{(l^2 + ll_0 + l_0^2) + t(l + 2l_0)}{3(l + l_0)}.$$

In the studies for design, referred to early in this paper, the analytic work was checked throughout by the graphic method wherever possible. This should always be done, both in designing and investigating cross-sections.

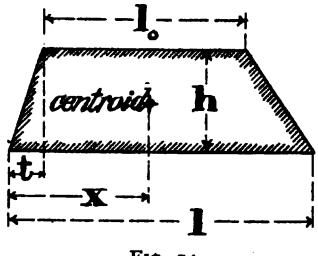


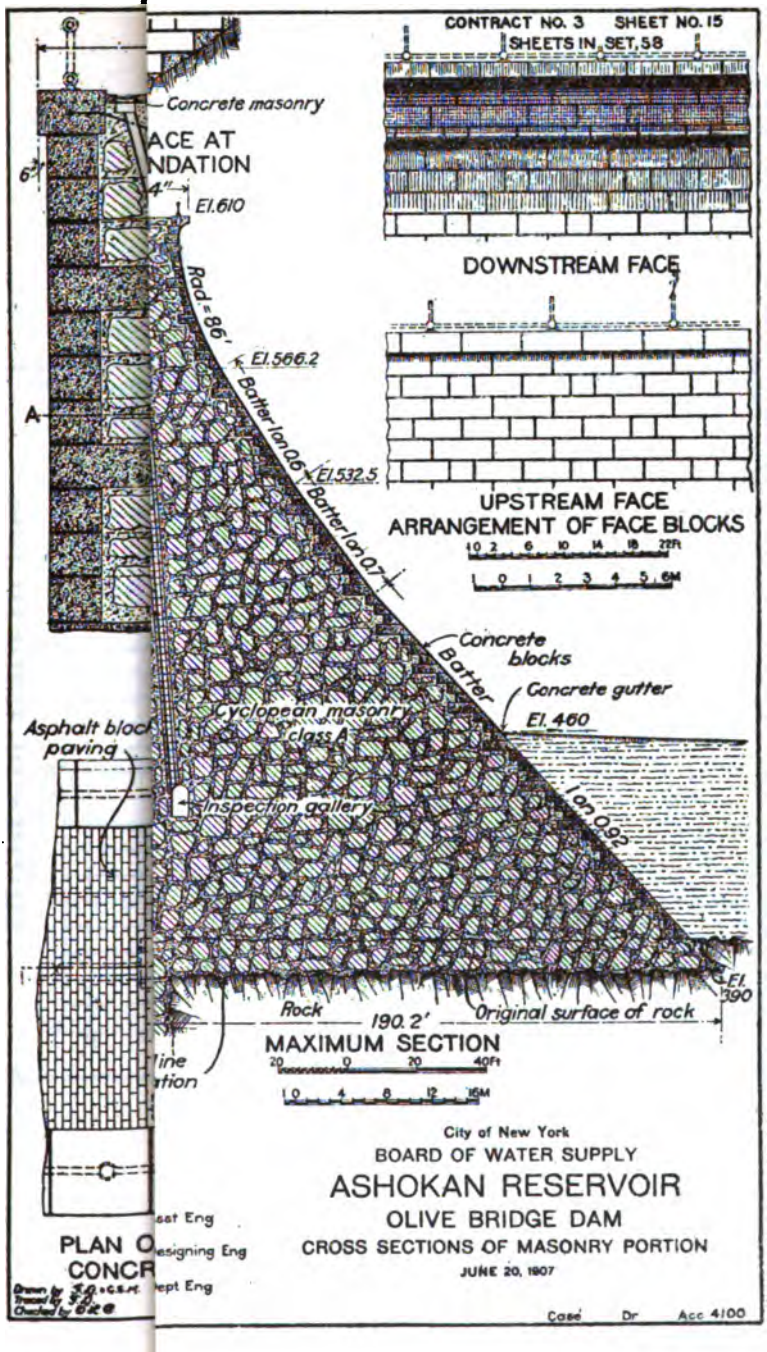
FIG. 14.

It should be stated here that, after a cross-section has been fixed upon for a given dam and the faces drawn to chosen batters and curves, the entire cross-section should be investigated as just outlined, so as to give the actual values for this final cross-section.

Again, in comparing different cross-sections, especially of different dams, by superimposing, their *water lines* should be made to coincide, and not their tops, for a fair comparison.

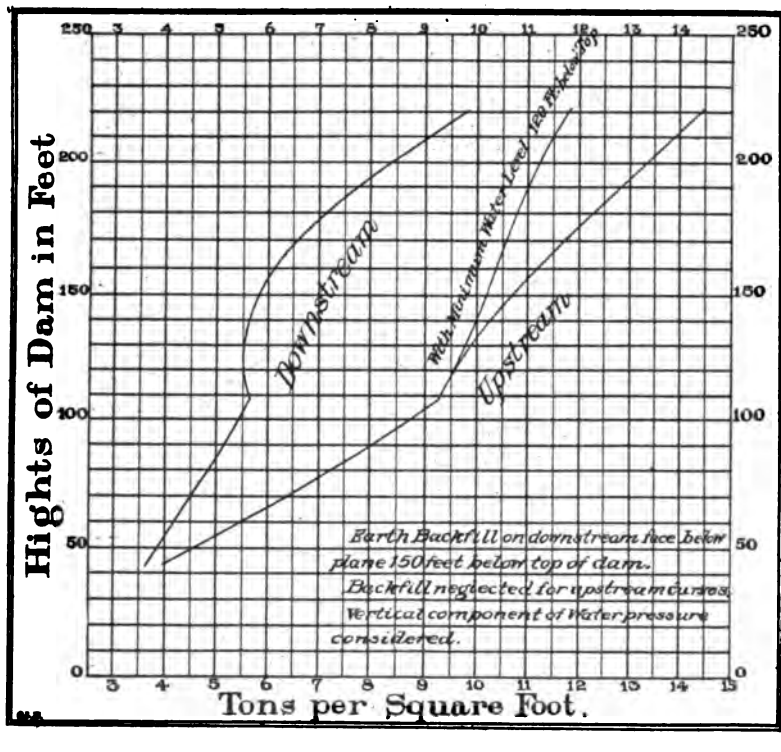
Plate No. I, from the contract drawings of Contract No. 3, let by the Board of Water Supply, shows the proposed cross-section of the Olive Bridge Dam, while Plate No. II gives curves of pressure intensities for this same cross-section. The studies for design of this masonry dam were conducted throughout the summer and fall of 1906 to the spring of 1907, and construction work is started.

JANUARY 1, 1908.


CONTRACT NO. 3 SHEET NO. 15
SHEETS IN SET, 58
Concrete masonry
ACE AT
NDATION
6'
4"
El. 610
Rad = 86'
El. 566.2
Batter 1on 0.6
Batter 1on 0.7
El. 532.5
A
DOWNSTREAM FACE
UPSTREAM FACE
ARRANGEMENT OF FACE BLOCKS
Concrete blocks
Batter
Concrete gutter
El. 460
Cyclopean masonry class A
Asphalt block paving
Inspection gallery
1on 0.92
El. 390
Rock
190.2'
Original surface of rock
MAXIMUM SECTION
City of New York
BOARD OF WATER SUPPLY
ASHOKAN RESERVOIR
OLIVE BRIDGE DAM
CROSS SECTIONS OF MASONRY PORTION
JUNE 20, 1907
PLAN O
CONCR
Case Dr Acc 4100

OLIVE BRIDGE DAM

Maxima Pressures on Horizontal Joints



DISPLACEMENT DIAGRAMS OF FRAMED STRUCTURES BY DEFLECTION ANGLES.

By MYRON S. FALK, Ph.D.

The usual and familiar graphical method of determining the displacements of the panel points of any structure employs the Williott diagram.

Under certain conditions, however, the Williott diagram fails to furnish an exact solution, particularly when the perpendiculars, which are erected at the ends of the lines denoting the strains in any member, intersect at an acute angle.

A graphical method which overcomes this difficulty makes use of the angular distortions occurring in the triangles forming the framed structure, together with the strains occurring in the various members. The changes in the angles of the structure may be found either by graphical or analytical methods; although the determination of these changes may involve more labor than the construction of the Williott diagram, the results obtained may also be used in the solution of other problems. For example, the values of the angular distortions furnish the means of determining the secondary stresses occurring in the members of a rigidly connected structure, due to the restraint occasioned when the members are not free to rotate about frictionless pins.

The Displacement Diagram.

Let the line ab, Fig 1, represent a truss member subjected to tension and assume the panel point a to have been displaced to a'. Let it be required to determine the final position of b, assuming that the member ab in addition to its direct stress of tension, also suffers an angular rotation $\Delta\varphi$ about a.

By the usual method of the Williott diagram, the new position of the line ab is first found at $a'b_1$. Since ab suffers an elongation of $\Delta l = b_1b_2$, the point b_1 will next be found at b_2. If the line $a'b_2$ is now rotated the amount $\Delta\varphi$ about a', the final position of b_2 will be

at b', and the total displacement of the point b from its original position will be given in amount and direction by the line bb'.

It is evident from Fig. 1, that the value of the line $b'b_2$ may be expressed by

$$b'b_2 = (l + \Delta l)\Delta\varphi, \tag{1}$$

but as the value of Δl is negligible when compared to l, it will be sufficiently accurate to write

$$b'b_2 = l\Delta\varphi = \rho. \tag{2}$$

The quantity ρ furnished a means of determining the displacement bb' by means of a separate diagram employing a pole O from which all displacements are measured, as in the case of the Williott diagram. If O (Fig. 2) represents the pole, the displacement of a

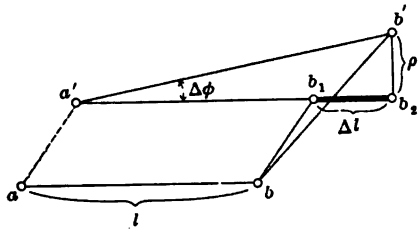


Fig.1

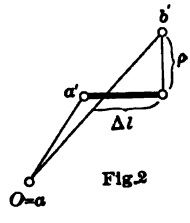


Fig.2

with regard to that pole is denoted by Oa', as found in Fig. 1. The line Δl is then drawn from a' parallel to ab to represent the elongation in that member, and the perpendicular ρ erected at the extremity of this line; Ob' will then represent the displacement of the point b when referred to the pole O.

The application of this method to a series of connected bars, is shown in Fig. 3; the broken line *abcde* . . . represents a system of connecting members of a framed structure, which are subjected to direct stresses of tension and compression only and which are attached to one another by frictionless hinges. Each member will be lengthened or shortened by an amount Δl and every angle between connecting members is assumed to be changed by an amount $\Delta\varphi$, which it will later be shown can be calculated. The displacement of the points of the broken line may be found graphically in the following manner: assume the direction of ab to be fixed, and the end a to be rigidly fastened, and let $O = a'$ (Fig. 4) represent the point a in the displacement diagram. The line

$+ \Delta l_1$ (Fig. 4) is then drawn parallel to ab (Fig. 3) to represent the displacement due to the change of length of that member, and since the member ab is in tension, the line Δl_1 must be drawn in the direction of a to b. If, on the contrary, this member were in compression, the direction of the line Δl_1 would have been drawn from b to a or downwards in Fig. 4.

Since the angular change at the panel point b is assumed to be known, the quantity ρ_b is determined from Eq. 2, as $l_2 \cdot \Delta\varphi_b$, and a

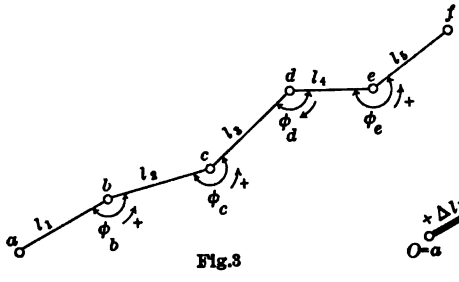


Fig.3

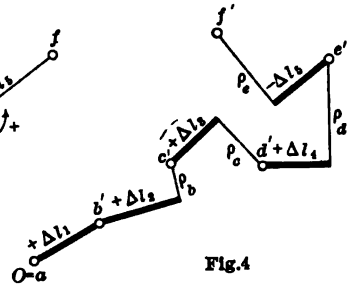


Fig.4

perpendicular of this length is erected at the end of Δl_1, the end of which, b', will denote the displaced position of b when measured from the pole O.

The rotation of any member towards the right is denoted (Fig. 3) by a positive sign on the arrow, while rotation to the left is denoted by a negative sign.

The positive sign before quantity Δl denotes a tensile strain, and a negative sign denotes a compression strain or shortening; for example, the member l_5 suffers compression.

Fig. 4 is continued by drawing successively $+ \Delta l_2$ at b' parallel to bc, and erecting a perpendicular at its extremity whose length is determined by

$$\rho_c = l_3 \cdot \Delta\omega_c = l_3(\Delta\varphi_b + \Delta\varphi_c).$$

The displacement of c is then found at c'; similarly that of d at d'; e at e' and finally f at f'. All these displacements are measured in the direction of the pole O to the primed letter.

It should be carefully noted that the quantity ρ does not merely depend on the change of angle at the point whose letter is its subscript, but that it depends upon the sum of the changes in every preceding angle, including the angle at the point in question.

Figs. 5 to 8 illustrate this point effectively. Let *abcdef*··· Fig. 5, represent a system of connecting members, precisely as in the case of Fig. 3, but let all the members lie in the same straight line. Assume that the only angular displacement occurs at *b*, the line *ab* remaining fixed in direction. The displaced position of *c* (Fig. 6) then becomes c_1; *d* is found at d_1; *e* at e_1, etc.

If Fig. 7 represents the displacement polygon, the pole *O* representing the position of *a*, then *Oc'* represents the displacement of *c* and *Od'* that of *d*; that is, due to the angular displacement at

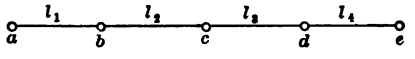


Fig.5

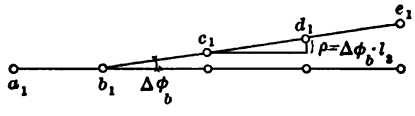


Fig.6

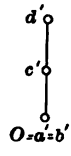


Fig.7

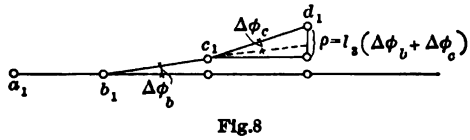


Fig.8

the panel point *b*, *d* is displaced not only the amount *Oc'* of the preceding panel point, but an additional amount represented by $c'd' = l_3 \cdot \Delta\varphi_b$. Consequently, if there is also an angular displacement $\Delta\varphi_c$ at *c*, as in Fig. 8, the displacement of *c* is found to be

$$\rho_c = l_3(\Delta\varphi_b + \Delta\varphi_c) = l_3 \cdot \Delta\omega_c.$$

The quantity $\Delta\omega$ will hereafter represent the sum of preceding $\Delta\varphi$'s.

Determination of the Angular Distortions of Triangular Frames.

The changes occurring in the angles of any triangular frame may be found either by algebraic or graphical methods. The following method will furnish algebraic equations of simple character.

Let *abc*, Fig. 9, represent a triangular frame in which the bar l_1 is subjected to tension and the bars l_2 and l_3 are subjected to compression. It is required to find the changes occurring in the angles φ_a, φ_b and φ_c.

Assuming the point a fixed in position and the line ab fixed in direction, the displacement diagram, Fig. 10, may be drawn, a' representing the pole from which all displacements are to be measured. The displaced position of b is then found at b', removed the distance $-\Delta l_3$ from a'. In order to locate c', the methods of the Williott diagram may be employed. Lines parallel respectively to ac and bc are drawn from a' and b' to represent the changes in length of those members. Perpendiculars are then erected at the

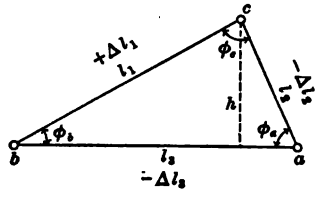


FIG. 9.

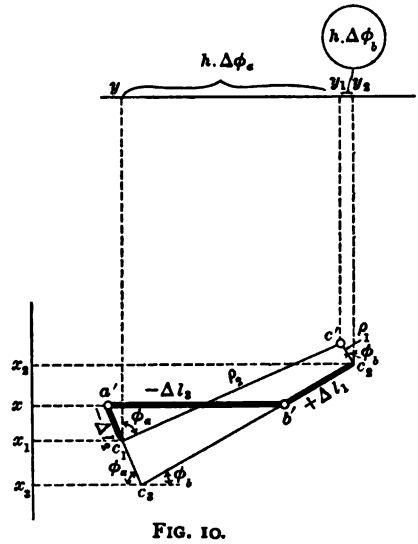


FIG. 10.

extremities of these two lines and their intersection at c' furnishes the displaced position of c. It is at once evident from the figure that the line c_1c' is the quantity ρ_3 of the preceding paragraph and that c_2c' is the quantity ρ_1. The following equations at once result:

$$c_1c' = \rho_2 = l_2 \cdot \Delta\varphi_a, \tag{1}$$

$$c_2c' = \rho_1 = l_1 \cdot \Delta\varphi_b. \tag{2}$$

Erect the line $x_3 \cdot x_1 \cdot x \cdot x_2$ at right angles to $a'b'$ in Fig. 10, and project upon it the points c_3, b', c_1 and c_2. The point c_3 is found at the intersection of the continuation of $+\Delta l_1$ and $-\Delta l_2$. Draw also a line $y \cdot y_1 \cdot y_2$ parallel to $a'b'$ and project upon this line the points c_1, c' and c_2. Erect also in Fig. 9 the perpendicular h upon the base ab.

From the similarity of triangles in Figs. 9 and 10, the following quantities may be found:

$$xx_2 : \Delta l_1 :: h : l_1 \quad \therefore xx_2 = \frac{\Delta l_1 \cdot h}{l_1}. \tag{3}$$

Similarly

$$xx_1 = -\frac{\Delta l_2 \cdot h}{l_2}, \tag{4}$$

$$xx_3 = -\frac{\Delta l_3 \cdot h}{l_3}. \tag{5}$$

Also

$$yy_1 = c_1c' \cdot \sin \varphi_a = \rho_2 \cdot \sin \varphi_a = l_2 \cdot \Delta\varphi_a \cdot \sin \varphi_a = h \cdot \Delta\varphi_a \tag{6}$$

$$y_1y_2 = h \cdot \Delta\varphi_b. \tag{7}$$

Combining Eqs. (4) and (5) and Eqs. (3) and (5)

$$x_1x_3 = -\frac{\Delta l_3 \cdot h}{l_3} + \frac{\Delta l_2 \cdot h}{l_2}, \tag{8}$$

$$x_3x_2 = -\frac{\Delta l_3 \cdot h}{l_3} + \frac{\Delta l_1 \cdot h}{l_2}. \tag{9}$$

Combining Eqs. (6) and (7)

$$yy_1 + y_1y_2 = h \cdot \Delta\varphi_a + h \cdot \Delta\varphi_b = c_1c_3 \cdot \cos \varphi_a + c_3c_2 \cdot \cos \varphi_b, \tag{10}$$

$$c_1c_3 = \frac{x_1x_3}{\sin \varphi_a} \quad \text{and} \quad c_3c_2 = \frac{x_3x_2}{\sin \varphi_a}. \tag{11}$$

Therefore, by substitution

$$\begin{aligned} h \cdot \Delta\varphi_a + h \cdot \Delta\varphi_b = &\left(-\frac{\Delta l_3 \cdot h}{l_3} + \frac{\Delta l_2 \cdot h}{l_2}\right) \cot \varphi_a \\ &+ \left(-\frac{\Delta l_3 \cdot h}{l_3} + \frac{\Delta l_1 \cdot h}{l_1}\right) \cot \varphi_b. \end{aligned} \tag{12}$$

But since the sum of the angles of any triangle at all times equals 180 degrees,

$$\Delta\varphi_a + \Delta\varphi_b + \Delta\varphi_c = 0, \tag{13}$$

$$\Delta\varphi_a + \Delta\varphi_b = -\Delta\varphi_c. \tag{14}$$

Therefore, substituting this value in Eq. (12),

$$\Delta\varphi_c = \left(\frac{\Delta l_3}{l_3} - \frac{\Delta l_2}{l_2}\right) \cot \varphi_a + \left(\frac{\Delta l_3}{l_3} - \frac{\Delta l_1}{l_1}\right) \cot \varphi_b. \tag{15}$$

And similarly there may be obtained:

$$\Delta\varphi_a = \left(\frac{\Delta l_1}{l_1} - \frac{\Delta l_2}{l_2}\right)\cot\varphi_c + \left(\frac{\Delta l_1}{l_1} - \frac{\Delta l_3}{l_3}\right)\cot\varphi_b, \qquad (16)$$

$$\Delta\varphi_b = \left(\frac{\Delta l_2}{l_2} - \frac{\Delta l_1}{l_1}\right)\cot\varphi_c + \left(\frac{\Delta l_2}{l_2} - \frac{\Delta l_3}{l_3}\right)\cot\varphi_a. \qquad (17)$$

Equations (15), (16) and (17) enable the change $\Delta\varphi$ of any angle of a triangle to be calculated; but this equation may be simplified for actual use by replacing the changes in length Δl, by the unit stresses p in the members, and the coefficient of elasticity E, if the coefficient E is constant for all the members of the structure. Should the structure be composed of several kinds of material involving different values of E, it should be observed that the following equations cannot be used.

It is known that

$$\Delta l = \frac{p \cdot l}{E} \qquad (18)$$

if p = stress of any member in pounds per square inch,
l = length of the member in inches,
E = coefficient of elasticity in pounds per square inch.
By substitution of Eq. (18) in Eq. (15) there is found

$$E \cdot \Delta\varphi_c = (p_3 - p_2)\cot\varphi_a + (p_3 - p_1)\cot\varphi_b \qquad (19)$$

and further

$$E \cdot \Delta\varphi_b = (p_2 - p_1)\cot\varphi_c + (p_2 - p_3)\cot\varphi_a, \qquad (20)$$

$$E \cdot \Delta\varphi_a + (p_1 - p_2)\cot\varphi_c + (p_1 - p_3)\cot\varphi_b. \qquad (21)$$

The quantity ρ may then easily be found from Eqs. 19 to 21 since

$$\rho = l \cdot \Delta\varphi = \frac{l(E \cdot \Delta\varphi)}{E}. \qquad (22)$$

EXAMPLE.

The preceding method may be applied to the truss * shown in Fig. 11; the span length is 240 feet, and the other dimensions are shown in the figure. The unit stresses p, and the changes of length Δl are also indicated for each member. The end a is the fixed

*The deflections of this truss, by the method employing the Williott diagram, are given in Burr & Falk's Influence Lines, page 168.

end; the point i moves on a horizontal roller-bed. E is assumed to be 28,000,000 pounds per square inch.

In Fig. 11 the cotangents of the various angles are shown for each angle, as well as the calculated values of $E \cdot \Delta\omega$ and ρ.

As in the case of the Williott diagram, a less bulky displacement figure will result, if at first instead of assuming the end point as the pole and fixed in position, a point more likely to receive the maximum displacement be chosen as the pole. In this case e is assumed as fixed in position and the member de as fixed in direction. The first ρ to determine is therefore that corresponding to the member cd for the angular change occurring at d. This angular change is, however, the sum of the changes in the three angles cdk, kdl and lde.

The angular change in cdk is found from the triangle cdk; substituting in Eq. 19, there is found

$$E \cdot \Delta\varphi_{cdk} = (-1000 - 12000)0 + (-1000 - 7400)1.2 = -10{,}080 \text{ pounds per square inch.}$$

This quantity is written within the angle cdk in Fig. 12.

Similarly

$$E \cdot \Delta\varphi_{kdc} = (-10{,}000 - 7{,}400)0.64 + (-10{,}000 - 1000)0.128 = -12{,}528 \text{ pounds per square inch.}$$

$$E \cdot \Delta\varphi_{ldc} = (4800 - 1000)1.33 + (4800 - 12000)0.75 = -350 \text{ pounds per square inch.}$$

The sum of these quantities $= E \cdot \Delta\omega_{de} = -22958$ and is also indicated in the figure.

In precisely the same way, the changes occurring at c and d may be found; and subsequently $E \cdot \Delta\omega_{eb}$ and $E \cdot \Delta\omega_{ba}$ may be evaluated. It should be particularly noted that

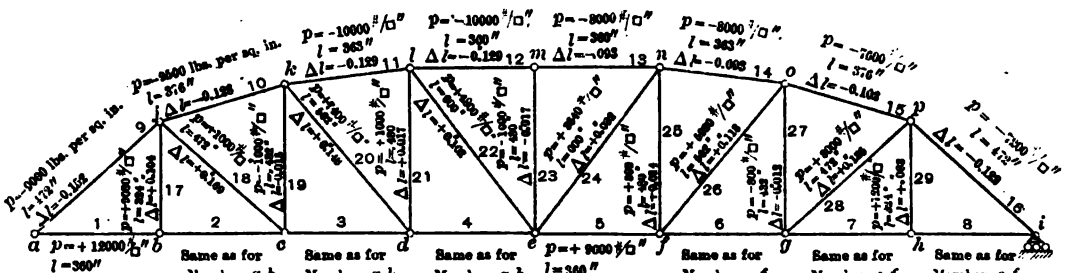
$$E \cdot \Delta\omega_{cb} = E \cdot \Delta\omega_{dc} + E \cdot \Delta\varphi_{bcf} + E \cdot \Delta\varphi_{fck} + E \cdot \Delta\varphi_{kcd};$$

that is, for the determination of ρ, the calculated quantity $E \cdot \Delta\omega$ at any point involves the similar quantity at all preceding points, as well as the quantities $E \cdot \Delta\varphi$ at the point in question.

Similar quantities are then also found at e, f, g and h; for examples

$$E \cdot \Delta\omega_{ef} = -5050 - 11{,}100 - 8880 - 4050 = -29080.$$

$$E \cdot \Delta\omega_{fg} = -29080 - 280 - 10000 - 8000.$$


p=-9000 lbs. per sq. in.
l=173"
Δl=-0.156
p=-9600 lbs. per sq. in.
l=57.6"
Δl=-0.138
p=-10000 #/□"
l=365"
Δl=-0.139
p=-10000 #/□"
l=300"
Δl=-0.139
p=-8000 #/□"
l=300"
Δl=-0.093
p=-8000 #/□"
l=365"
Δl=-0.093
p=-7600 #/□"
l=57.6"
Δl=-0.138
p=-7600 #/□"
l=173"
Δl=-0.156
p=+12000 #/□"
l=300"
Same as for Member a b
Same as for Member a b
Same as for Member a b
p=+9000 #/□"
l=360"
Same as for Member e f
Same as for Member e f
Same as for Member e f
a
b
c
d
e
f
g
h
i
k
l
m
n
o
p

Since

$$\rho = l \cdot \Delta\omega = \frac{l \cdot (E \cdot \Delta\omega)}{E},$$

and since in this problem l is uniformly 360 inches and E 28,000,000, the quantities ρ may at once be found from $E \cdot \Delta\omega$ by a simple multiplication; the calculated values are shown in Fig. 12.

The displacement diagram Fig. 13 may then be drawn as follows: e' represents the pole or fixed point; $e'd''$ represents the elongation in ed, the direction of ed being assumed fixed. At d'' a line parallel to dc is drawn, indicating the change of length Δl_{dc} and at its extremity the perpendicular $\rho = 0.295$, to locate c'. At c' a line parallel to cb is drawn with a length of 0.154 inch to denote the strain Δl_{cb} in that member, and at its extremity the perpendicular ρ, with a length of 0.403 inch to locate b'. Continuing this form of construction, the points a', f', g', etc., may be found. The figure thus drawn would apparently show a displacement of the actual fixed point a of an amount represented by the line connecting d'' and a'. Since it is a however, and not e, which is fixed, the pole of the displacement diagram is really $a' = a''$. The rotational displacement figure which is used in similar cases to, complete the Williott diagram is then employed * to represent the actual displacements of any point. Since the point i can move only horizontally, the line $i'i''$ is drawn horizontally till it intersects at i'' the line drawn through a'' at right angles to the line ai. Upon this line $a''i''$ there is then drawn a figure of the truss every line at right angles to the original figure. The final displacement of any point is then measured from the second primed letter to the first primed letter.

The final and true displacement of e is for instance $e''e'$; that of h, $h''h'$. It is evident from the figure that the displacement of the point i to the right is measured by the sum of the elongation of the lower panel members.

The Deflection Polygon.

If the vertical displacements of the lower chord panel points only are desired, the additional polygon $a_1b_1c_1d_1e_1$, etc., may be constructed by obtaining the intersection of horizontal lines through

* For the detailed description of this rotational diagram, see Burr and Falk's Influence Lines, page 166.

the primed points of Fig. 13 with verticals dropped from the panel points of Fig. 12, and connecting the points a_1 and i_1 with a straight line. The intercepts between this line and the broken line $a_1 . b_1 . c_1 \ldots$ represent the vertical displacements.

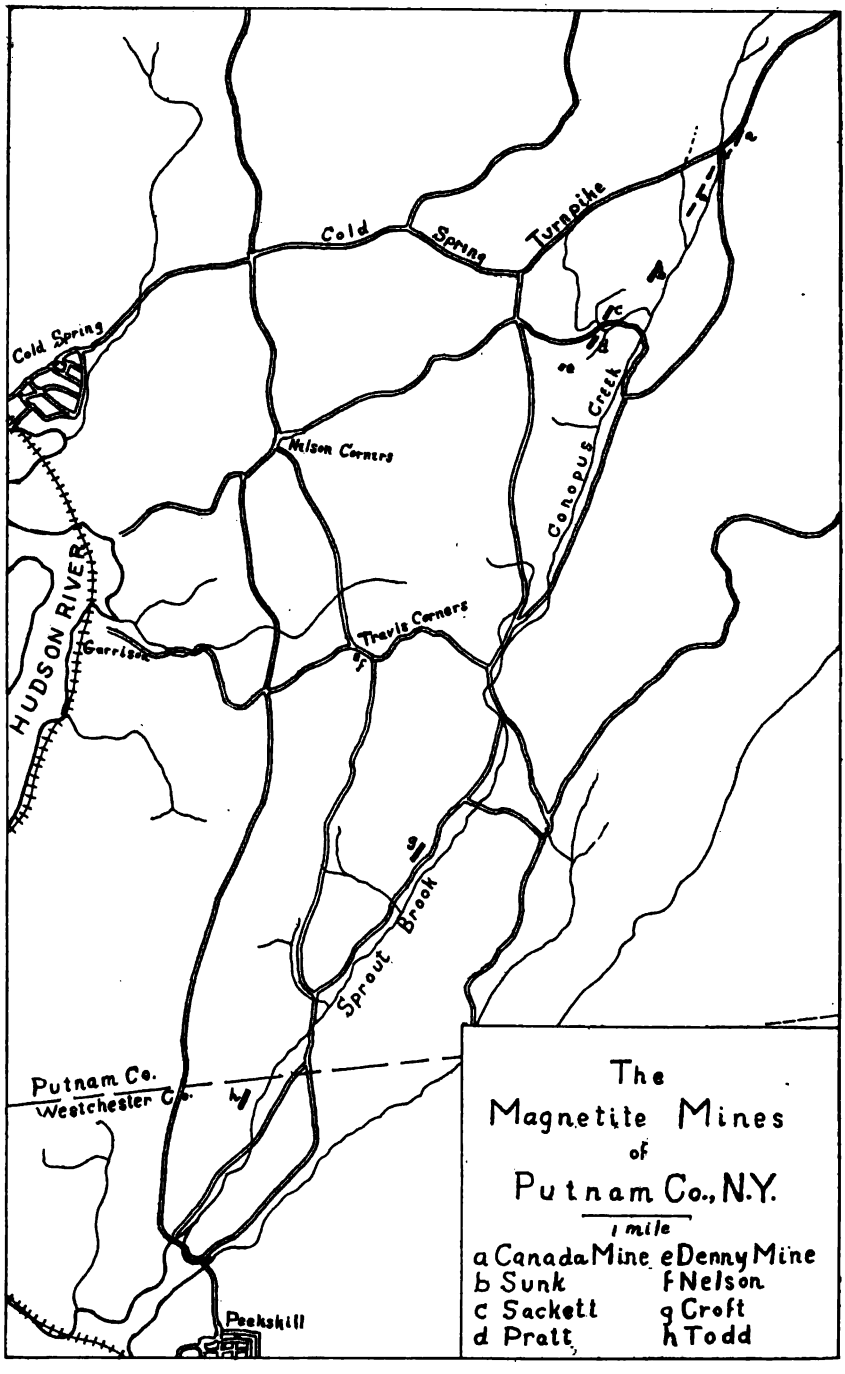
It is evident that this last figure resembles closely the ordinary moment polygon, drawn for determining the moments existing in a horizontal beam carrying fixed vertical loads. Consequently it would appear to be feasible to draw such a deflection bolygon by precisely the same methods employed for any ordinary moment polygon; it is only necessary to determine by some means the proper loads at the various panel points. The methods for determining such loads will be shown in a later paper.

THE MAGNETITE BELTS OF PUTNAM COUNTY, N. Y.

By C. A. STEWART.

The origin of the magnetite lenses in the gneisses of the Highlands of New York and New Jersey has been the subject of much discussion. Older observers, who believed all banded gneiss to be sedimentary, unhesitatingly classed these ores as metamorphosed sediments. More recent studies, however, have convinced us of the important part played by igneous rocks in forming many of the banded gneisses of this area. With this new conception has come a feeling that the genesis of the magnetite beds is still an open question. This paper undertakes a description of the wall rocks of a few closely associated deposits. It is not intended to support any theory of origin, but merely to set forth certain facts that may be of value in the future. It is believed that, as careful study of the Highlands progresses, detailed descriptions of most of these deposits will enable us to separate them into several types, each characterized by features the true significance of which does not appear when they are considered as local developments.

Dr. Chas. P. Berkey has done the most recent work in the Highlands of New York and the writer is indebted to him and to Prof. J. F. Kemp for help in preparing this paper. Upon Dr. Berkey's publication in N. Y. State Museum Bull. 107, the following summary of the geology is based. The prevailing types of rock are banded hornblendic, micaceous and quartzose gneisses. Conformably interbedded with these are several varieties of schist, rarer gneisses, limestones and quartzites. The limestones are often in more or less lenticular form, from ten to thirty feet thick, entirely surrounded by the gneiss. The quartzites are similar to the limestones in occurrence, but are more extensive. The whole series is folded, crumpled, faulted, crushed, sheared, injected, and intensely modified by recrystallization, but through it all it retains the fundamental associations and essential character of a sedimentary series. It contains gneisses, schists, granites, diorites, and gabbros that are undoubtedly metamorphosed equivalents of intrusions in the orig-


Cold Spring Turnpike
Cold Spring
Nelson Corners
Canopus Creek
HUDSON RIVER
Garrison
Travis Corners
Sprout Brook
Putnam Co.
Westchester Co.
Peekskill
The Magnetite Mines of Putnam Co., N.Y.
1 mile
a Canada Mine
b Sunk
c Sackett
d Pratt
e Denny Mine
f Nelson
g Croft
h Todd

inal sediments. In many cases the intrusive masses are sheet-like, and parallel the banding of the other gneisses. Pegmatite streaks are abundant, both as off-shoots of larger igneous masses, and as local developments.

In Putnam County the magnetite deposits fall into two belts, parallel to the topographical and geological features of the district. One beginning just north of the Cold Spring turnpike, about seven miles east of Cold Spring, can be followed three and a half miles to the southwest. The other lies along Sprout Brook, extending seven and a half miles north of Peekskill. The ore-bodies are lenses or beds, lying parallel to the foliation of the country rock, and pinching out at the ends. The ore is hard, massive magnetite, with minute, irregular inclusions of feldspar, quartz, hornblende, apatite, and other silicates common to the wall rocks. (See Fig. 1).

FIG. 1. Ore from Canada mine. *Q* is quartz; *P*, plagioclase; *O*, orthoclase; *Hb*, hornblende; *M*, magnetite. (× 20.)

According to B. T. Putnam * it varies from .12 to .66 per cent. in phosphorus. No analyses of sulphur or titanium are available, but the latter is not high enough to bring the ore in the class of the refractory, titaniferous magnetites. In one case calcite forms the gangue, while elsewhere the country rock is the barren admixture with the ore.

THE CANADA MINES.

Like all of the workings in this district, the Canada mines have been abandoned for over twenty years. At present they are repre-

* *Tenth Census of U. S.*, XV, pp. 88, ff.

sented by a series of pits running northeast along Cold Spring turnpike near Conopus Creek. Here the strike of the gneiss is N. 40° E. and the dip 60° to the southeast. Just east of the creek an open cut indicates the removal of a bed of ore six to eight feet thick, that continued with many pinchings and swellings a quarter of a mile northeasterly along the road. The workings to the southeast are small ditches, evidently largely prospects. The ore formed a solid band of magnetite grading gradually on both sides into the country rock, which is a gray gneiss. In its leaner portions it alternates with the gneiss in bands as sharply defined as veins and always parallel to the foliation. This gneiss is chiefly albite, quartz, and orthoclase, with varying amounts of magnetite.

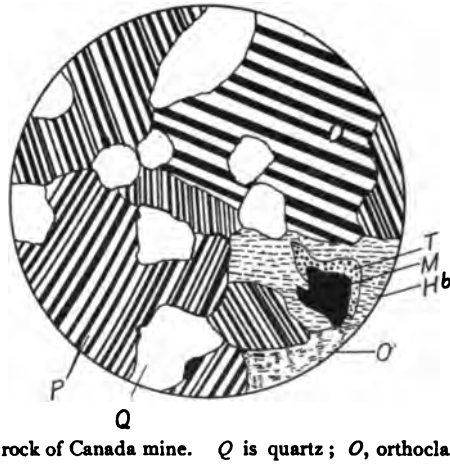


FIG. 2. Wall rock of Canada mine. *Q* is quartz; *O*, orthoclase; *P*, plagioclase; *Hb*, hornblende; *M*, magnetite; *T*, titanite. (× 60.)

The feldspars are nearly rectangular in shape, average half a millimeter in length, and usually include the quartz. Besides being included in the feldspars, quartz is found in small rounded grains. When the feldspar does not surround the quartz, it is usually xenomorphic in respect to it. In no case was quartz found in elongated continuous masses that generally result from its recrystallization. Magnetite occurs both as minute grains scattered throughout the rock, and in parallel stringers that reach out into the interstices between the feldspars, partly surrounding them. (See Figs. 2 and 3.) With it are associated titanite, apatite, zircon, small shreds of biotite, and bluish to yellow pleochroic hornblende. The dark silicates and titanite are not common to

other areas of the rock. All the minerals show signs of intense strains. Wavy extinction is universal, and the plagioclase in particular gives beautiful examples of bent and faulted crystals. In many cases magnetite has penetrated the cracks of a broken feldspar and completely surrounded the fragments. Mortar structure is rare, and the quartz grains in only a few instances show comminution along their borders. Although the features as outlined above predominate, the relative amounts of the minerals are variable, and there are isolated areas in which excessive granulation has taken place. Even in these cases there are some well-developed feldspar crystals xenomorphic on the quartz.

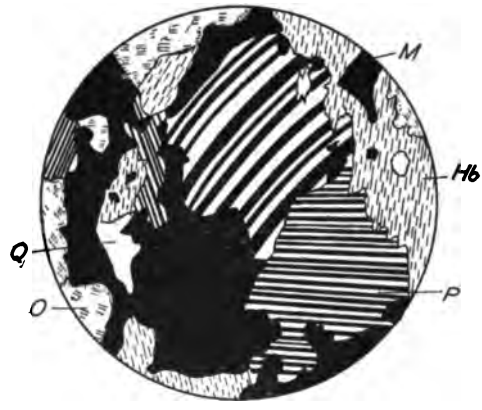


FIG. 3. Wall rock of Canada mine. *Q* is quartz; *O*, orthoclase; *P*, plagioclase; *M*, magnetite; *Hb*, hornblende. (× 60.)

In the foot-wall of one of the more northerly pits there is a stringer of pegmatite that carries great quantities of pyrite. It holds for only thirty feet, and pegmatites do not seem to be prominent features of this deposit.

THE SUNK OR STUART MINE.

The Sunk or Stuart mine is about two miles southwest from the main pit of the Canada. The workings are now much caved, but the bed evidently ran along the eastern flank of a steep hill of gneiss, and was parallel to the foliation, which strikes N. 45° E. and dips eighty degrees to the northwest. According to Putnam* the ore was mined for about 700 feet along the strike and 300 feet in

* *Loc. cit.*

depth, and varied from eight to ten feet in thickness. The gneiss forming the walls and gangue is in general the same as that of the Canada. In some cases excessive granulation is present, and the feature of feldspar moulded on quartz is not always extensively developed. One specimen of the foot-wall showed a predominance of orthoclase. A section cut adjacent to a pegmatite streak found in the dump is remarkable for what appear to be two varieties of feldspar. Rounded grains of plagioclase about half a millimeter in diameter are present in subordinate amount. The greater part of the feldspar is in large masses forming a sort of ground mass for the other minerals, and showing a flecked intergrowth of two differently oriented feldspars, neither of which has definite polysynthetic twinning.

Just north of the pit of this mine an old tunnel has exposed a ten-foot band of a definite sedimentary type that is interbedded with the gray gneiss in exactly the same relation as the ores. This rock is white with streaks of reddish brown. The white is quartz and albite varying from rounded grains to sharply interlocking angular masses. The brown streaks are a mixture of garnet, calcite, and clinozoisite. The garnet is the most abundant. It never has definite crystal outline, but sends out branching processes reaching into the interstices of the feldspars and quartz, partially surrounding them.

The Pratt and Sacket Mines.

The Pratt and Sackett mines are a quarter of a mile southwest of the Sunk and evidently form a single bed. The Sackett is just north of the road, while the Pratt is about sixty-five yards south of it. The strike of the gneiss and the parallel position of the ore are the same as in the mines described above. The ore in the Sackett is 5 feet wide and was worked 75 feet in depth and 175 feet along the strike. The Pratt bed is from 2 to 4 feet wide and 300 feet long. Excessive caving and weathering in the Sackett prevent careful study of the wall rock, which shows in the Pratt as gneiss similar in general appearance to that of the Sunk. Streaks of pegmatite cut across the banding, including fragments of both ore and wall rock, and emphasizing the general relationship of pegmatite younger than the ore bodies. Microscopically the gneiss is identical with that of the Canada mines, xenomorphic feldspar and granular quartz being very prominent.

The Denny Mine.

The Denny mine is at the top of a hill a quarter of a mile southwest of the Pratt. The abandoned workings indicate three pods, or lenses, the largest of which is 75 by 100 feet, all lying in the same line of strike. The foot-wall of the main pit is pegmatite, which cuts across the gneiss, and includes fragments of it and the magnetite. In general the gneiss is similar to the type prevailing throughout this belt, though it contains little quartz and somewhat unusual developments of hornblende and biotite.

The finer grained pegmatite consists of microcline, albite, orthoclase, quartz, chlorite, magnetite, and titanite. The feldspars, though occasionally granular, are more often large masses up to two millimeters in diameter. They show all manner of intergrowths — orthoclase, plagioclase and microcline — each including the others, as well as differently oriented particles of its own species. The quartz is usually concentrated in streaks or flat lenses three millimeters or more long and about half as wide. Its general relation to the rest of the rock is more that of the last product of crystallization than is shown by the quartz of any of the sections described above.

The Nelson Mine.

The Nelson mine is about 100 feet south of the road at Travers Corners, three and a half miles southwest of the Denny mine. It lies somewhat west of the strike of the belt described above, but shows similar relationships. The gneiss here strikes N. 55° E. and dips 45 degrees to the southeast. A shaft sunk along the dip is now filled with water so that no ore is visible. A resident in this vicinity said that the workings were abandoned when ore in paying quantities was not found after sinking the shaft ninety feet. The magnetite in the dump is uniformly parallel with the banding of the gneiss, which is cut by pegmatite. In mineralogy the wall-rocks differ from the Canada type in carrying an excess of quartz, and much orthoclase, although albite is present in considerable amount. A section near a pegmatite streak shows the quartz in large masses rolled out parallel to the foliation.

The Croft Mine.

The Croft and Todd mines are east of the strike of the Canada-Pratt belt and differ from those mines in important particulars. The Croft is on the eastern flank of a steep hill, just west of Sprout

Brook, about five miles northeast of Peekskill. The gneiss here strikes northeast and dips 50 degrees to the southeast, with a slight pitch to the southwest. The old workings are represented by an open cut extending 200 feet along the strike, now filled to within thirty feet of the surface by caving. The ore was apparently about ten feet wide. Several pillars have been left near the top of the cut. These show gneiss and parallel bands of magnetite that contained much pyrite and hence has weathered to a rotten, rusty material so that it is possible to break out slabs of the less ferruginous gneiss as regularly as fragments of shale. This gneiss is largely orthoclase and hornblende, with subordinate amounts of microcline, albite, biotite, titanite, and pyrite. No quartz or zircon was found. The orthoclase appears in large anhedra, while the other feldspars are in small (three-tenths of a millimeter) grains. The hornblende is muddy, olive green, in parallel bands, nearly always xenomorphic in respect to the feldspar, and often associated with pyrite. This rock shows very little bending, crushing or secondary growth. A thin section of regularly banded rock from the dumps resembled the above, if we except the addition of magnetite, a few grains of quartz, and ten or more isolated sharply defined grains of calcite.

The Todd Mine.

The Todd mine is west of Sprout Brook, three miles southwest of the Croft. The old workings indicate a nearly vertical bed of ore 150 feet long and three to four feet thick. According to Putnam * the eastern wall at the south of the mine is limestone which cuts across the ore about 75 feet to the north. None of this wall-rock can now be seen in place, but the dump shows that the gangue is chiefly limestone.

The western wall of the pit is a green schist consisting largely of shreds and patches of yellowish green hornblende with great secondary growths of actinolite forming long fibrous aggregates. The rest of the rock is a mass of very finely comminuted mineral, low in relief and double refraction — probably feldspar from the amount of decomposition. Ilmenite is in isolated grains invariably surrounded by remarkable growths of leucoxene. Twenty-five yards to the west this schist is replaced by a coarse-grained gneiss of feldspar and hornblende with some fibrous secondary growths.

* *Loc. cit.*

The former is chiefly orthoclase, largely altered to sericite and kaolin. Plagioclase occurs as small, rounded grains, half a millimeter or less in diameter. Ilmenite, leucoxene, and apatite are accessories. Shearing of this gneiss has probably produced the schist of the wall rock. Still further west the gneiss contains quartz in elongated masses, and the hornblende is replaced by light green mica. Two feldspars are still present, — the large masses packed with sericite, kaolin, and chlorite; and small, rounded grains, fresh and showing either polysynthetic twinning or microcline structure. (See Fig. 4.) Needles of rutile (?) are

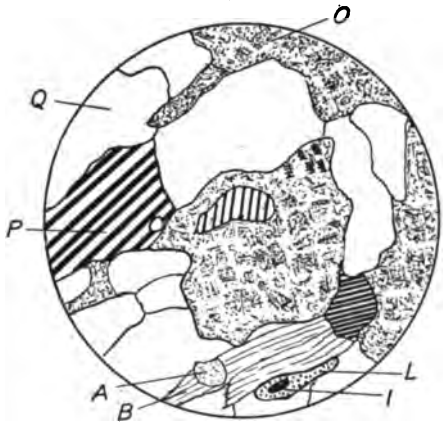


FIG. 4 Granitic gneiss west of Todd mine. *Q* is quartz; *P*, plagioclase; *O*, sericitic feldspar; *B*, biotite; *A*, apatite; *I*, ilmenite; *L*, leucoxine. (× 20.)

regularly arranged in some of the mica. Just east of the probable location of the limestone wall is a gneiss mineralogically similar to the schist on the west, but with important developments of epidote. The limestone in the dump is pure white and densely crystalline. It carries cubes of pyrite, and considerable epidote, but except an occasional light-colored pyroxene, is free from other silicates. Epidote is abundant in the old ore piles.

DISCUSSION.

The deposits of the Canada-Denny belt evidently form a series of lenses lying along a nearly straight line that is parallel to the strike of the country rock. They are essentially alike both in their relations to the wall-rocks and in the nature of these walls, and may reasonably be considered to have a common origin. The

complete recrystallization prevailing in this region and the lack of chemical analyses seriously impair the value of any attempt to demonstrate the sedimentary or igneous nature of the gneisses forming the wall-rocks. The evidence supporting a theory of igneous origin is largely negative. These rocks preserve no undoubted traces of original clastic structures; they show none of the aluminous minerals that usually result from the recrystallization of a highly feldspathic sediment; and they vary no more in mineralogy than would be expected in an eruptive subject to long-continued surficial decay, followed by deep-seated recrystallization. Any view of their igneous origin must consider the quartz as largely a metamorphic product, for in very few cases does it occur in forms that can be referred to the masses crystallizing from a magma.

On the other hand, complete absence of fragmental structure and of aluminous minerals characterizes certain feldspar-graphite schists that seem undoubtedly of sedimentary origin. Moreover, in spite of the great importance of banding original in intrusive rocks and induced in them by metamorphism, it is impossible to see the marvelously regular banding of these gneisses without feeling the force of the arguments advanced by so many of the earlier observers for the sedimentary origin of both the gneiss and the included ore lenses. This is especially true when we find this banding strongly suggesting interbedding with an unmistakable sediment as at the Sunk mine. If the wall rocks were formerly an arkose, the present relations of the quartz and feldspar can be explained by assuming that the feldspar, being more finely divided, recrystallized first and thus included the quartz grains. In a few cases the quartz may have been very fine or the metamorphic forces of unusual intensity and in this way may have been formed the larger elongated quartz masses that are occasionally seen. The striking association of the hornblende with the magnetite may be the result of recrystallization of a mixture of iron and the aluminous decomposition products of the feldspar. Possibly in this way we can account for the excess of alumina that would be left by a slight leeching of the alkali from the feldspar grains of an arkose.

Even if the sedimentary nature of the wall-rock were proven, there would still be a variety of ways in which to explain the origin of the ores. The absence of appreciable quantities of sulphides or of metamorphic lime minerals, and the presence of the heavy minerals zircon, titanite and apatite in invariable association with the magne-

tite would give great weight to the time-honored conception of concentration of magnetic sands. If we consider the associated gneisses as igneous rocks, we can refer the ore to the results of segregation or replacement; or it may be the metamorphosed equivalent of a portion of an old bed of iron ore caught up by an intrusive in the same way as certain limestone lenses in the Highlands of New Jersey. In any event, two points are certainly established in regard to the ore bodies of this belt. They are earlier than the pegmatites, as is shown distinctly in the Pratt and Denny mines. They have the same rock in both foot and hanging walls and are therefore not to be referred to a well-defined contact zone.

The deposit of the Todd mine differs from the above in the presence of limestone, in the character of the associated gneiss, and the lack of banding. Limestone cutting across this ore can be accounted for by postulating a lens of limestone partially replaced by iron — probably as sulphide — later changed to oxide. The accompanying gneiss is massive and strongly suggestive of igneous origin. Its peculiar structure of two feldspars, one including the other, was seen in well-defined pegmatite areas at the Sunk and Denny mines. It may be an acid intrusion of aqueo-igneous nature that furnished the solutions acting in the replacement process.

The Croft mine, although falling in the line of strike of the Todd, seems to be in no way connected with it. It resembles the Canada belt in regularity of banding, but differs from all the other deposits of this district in abundance of pyrite and in character of the wall-rock. The facts at hand are much less suggestive than those observed in the other deposits. The Nelson mine is similar to the Canada-Denny belt.

Thus in this limited district we have two distinct types of deposits. One suggests replacement of a limestone lens, possibly connected with the action of an acid intrusive. The other, free from traces of limestone and obviously older than the pegmatites, is in a rock of doubtful origin, but has no features that cannot be accounted for by considering it a metamorphosed bed of magnetite sand. Mr. A. C. Spencer * has traced a connection between the pegmatites and the magnetite lenses of the New Jersey Highlands. Professor Kemp † has shown the probability of certain magnetites

* Ann. Rpt. State Geologist of N. J. for 1904, pp. 249–252.

† *Trans. A. I. M. E.*, February, 1897.

of the Adirondacks being the result of the intrusion of basic eruptives. It therefore seems probable that the pre-Cambrian magnetite ores, although resembling each other in shape and having in a large way similar associations, are of various origins. In a complex series of sediments abundantly injected with many kinds of intrusives and afterwards intensely metamorphosed, it is but reasonable to suppose that sedimentation, segregation, chemical precipitation, replacement and contact action have all done, in various places, some work in concentrating so widely distributed a metal as iron.

GASES: TABLES AND CONSTANTS.

By GEORGE C. STONE.

Having recently had occasion to make a number of calculations of the performance of different gas-producers and the utilization of heat in different furnaces, I have been much annoyed by the failure of calculations to check with each other, on tracing the source of these errors I have found that they were due to variations in the published tables of the weights of gases. To prevent this annoyance I have calculated the following tables.

The weight of carbon in a given volume of CO_2, CO or CH_4 should be the same, but I have not found any published tables in which it is. As a basis I have taken the average weight of carbon in a liter of these gases as given in the Smithsonian tables and in Landholt and Börnstein. From this figure and the volumetric relations of the gases I have figured the weights of the other gases. These results agree very closely with the best experimental results.

The thermal capacities of gases have been calculated for each hundred degrees from 100 to 2,400 by Damour's formulas (Industrial Furnaces, American edition); for variations of temperature of less than 100 degrees the thermal capacities can be interpolated with sufficient accuracy for ordinary purposes.

I have added a table of conversion factors and one of the temperature of steam corresponding to different pressures.

All volumes have been calculated at 100° C., as in most cases water vapor has to be considered, and it appeared better to take a temperature at which large amounts of it could exist than to make the assumption that the volumetric relations would be the same at all temperatures. In general I have used metric weights and volumes but occasionally, as in the table of pressures and temperatures of steam I have mixed up the English and metric units. This has been done where in the ordinary course of work quantities or pressures were nearly certain to be given in English units.

I have ventured to publish these tables in the hope that they may save others from the useless task of trying to check results by the use of tables the units of which do not agree among themselves.

WEIGHTS OF GASES.

Gas.	Weight per Cu. Ft		Weight per Cu. M.	
	Lbs	Log.	Kilos.	Log
Air,	0.05907	$\bar{2}.771388$	0.94625	$\bar{1}.976006$
O	0.06520	$\bar{2}.814248$	1.04440	0.018866
N	0.05745	$\bar{2}.759290$	0.92025	$\bar{1}.963908$
H	0.004075	$\bar{3}.610129$	0.065275	$\bar{2}.814747$
CO	0.05705	$\bar{2}.756256$	0.91385	$\bar{1}.960874$
CO_2	0.08965	$\bar{2}.952550$	1.43605	0.157168
CH_4	0.03260	$\bar{2}.513218$	0.52220	$\bar{1}.717836$
C_2H_4	0.05705	$\bar{2}.756256$	0.91385	$\bar{1}.960874$
H_2O	0.036675	$\bar{2}.564311$	0.587475	$\bar{1}.768990$

WEIGHTS OF CONSTITUENTS.

Gas.	Oxygen.		Carbon.		Hydrogen.	
	K.	Log.	K.	Log	K.	Log.
1 cu. m. CO	0.52220	$\bar{1}.717837$	0.39165	$\bar{1}.592898$		
1 " " CO_2	1.04440	0.018865	0.39165	$\bar{1}.592898$		
1 " " CH_4			0.39165	$\bar{1}.592898$	0.13055	$\bar{1}.115776$
1 " " C_2H_4			0.78330	$\bar{1}.893928$	0.13055	$\bar{1}.115776$
1 " " H_2O	0.52213	$\bar{1}.717776$			0.06527	$\bar{2}.814686$
	Cu. M.		K.			
1 lb. C = CO	1.15816	0.063768	1.05838	0.024642		
1 lb. C = CO_2	1.15816	0.063768	1.66317	0.220936		

AIR COMPOSITION AND RATIOS.

Ratio.	By Weight.		By Volume.			
		Log.		Log.		Log.
O: N :: 1:	3.32670	0.522016	3.77555	0.576980		
N: O :: 1:	0.30059	$\bar{1}.477984$	0.26486	$\bar{1}.423020$		
O: Air :: 1:	4.32670	0.636157	4.77555	0.679023		
Air : O :: 1:	0.23112	$\bar{1}.363843$	0.20940	$\bar{1}.320977$		
N: Air :: 1:	1.30059	0.114141	1.26486	0.102043		
Air : N :: 1:	0.76888	$\bar{1}.885859$	0.79060	$\bar{1}.897957$		
	Air K.		Oxygen K.		Nitrogen K.	
1 cu. M. Air	0.94625	1.976006	0.21870	$\bar{1}.339849$	0.72755	$\bar{1}.861865$
1 cu. M. O	4.51881	0.655023	1.04440	0.018866	3.47441	0.540882
1 cu. M. N	1.19687	0.078049	0.27662	$\bar{1}.441892$	0.92025	$\bar{1}.963908$

GRAMS WATER IN SATURATED AIR.

Temp.	Per K.	Log.	Per M^3.	Log.	Temp.	Per K.	Log.	Per M^3.	Log.
0	3.75	0.574031	4.848	0.685583	16	11.13	1.046595	13.593	1.133312
1	4.03	0.605305	5.191	0.715269	17	11.86	1.074085	14.435	1.159402
2	4.32	0.635484	5.545	0.743866	18	12.64	1.101747	15.331	1.185570
3	4.64	0.666518	5.934	0.773324	19	13.46	1.129045	16.270	1.211377
4	4.98	0.697229	6.346	0.802464	20	14.33	1.156246	17.262	1.237093
5	5.34	0.727541	6.780	0.831211	21	15.25	1.183270	18.308	1.262638
6	5.71	0.756636	7.224	0.858747	22	16.22	1.210051	19.406	1.287944
7	6.13	0.787400	7.727	0.888017	23	17.24	1.236537	20.557	1.312960
8	6.56	0.816904	8.240	0.915913	24	18.32	1.262925	21.771	1.337884
9	7.02	0.846337	8.786	0.943803	25	19.47	1.289366	23.060	1.362865
10	7.51	0.875640	9.366	0.971569	26	20.68	1.315551	24.412	1.387595
11	8.03	0.904716	9.980	0.999113	27	21.95	1.341435	25.824	1.412029
12	8.58	0.933487	10.626	1.026357	28	23.29	1.367169	27.310	1.436318
13	9.16	0.961895	11.304	1.053244	29	24.70	1.392697	28.867	1.460405
14	9.78	0.990339	12.027	1.080172	30	26.18	1.417970	30.496	1.484242
15	10.43	1.018284	12.782	1.106607					

AIR REQUIRED TO BURN.

Gas.	K.	Log.	Cu. M.	Log.
1 K. C to CO	5.76893	0.761096	6.09662	0.785090
1 K. C to CO_2	11.53786	1.062126	12.19324	1.086120
1 K. CO	2.47240	0.393119	2.61284	0.417113
1 K. CH_4	17.30680	1.238217	18.28988	1.262211
1 K. C_2H_4	14.83439	1.171270	15.67703	1.195264
1 K. H	34.61360	1.539247	36.57976	1.563241
1 Cu. M. CO	2.25940	0.353993	2.38774	0.377987
1 Cu. M. CH_4	9.03761	0.956053	9.55097	0.980047
1 Cu. M. C_2H_4	13.55641	1.132144	14.32643	1.156138
1 Cu. M. H	2.25940	0.353993	2.38774	0.377987

OXYGEN REQUIRED TO BURN.

1 K. C to CO	1.33333	0.124939	1.27665	0.106073
1 K. C to CO_2	2.66667	0.425969	2.55330	0.407103
1 K. CO	0.57143	1.756962	0.54714	1.738096
1 K. CH_4	4.00000	0.602060	3.82996	0.583194
1 K. C_2H_4	3.42857	0.535113	3.28282	0.516247
1 K. H	8.00000	0.903090	7.65991	0.884224
1 Cu. M. CO	0.52220	1.717837	0.50000	1.698970
1 Cu. M. CH_4	2.08880	0.319897	2.00000	0.301030
1 Cu. M. C_2H_4	3.13320	0.495988	3.00000	0.477121
1 Cu. M. H	0.52220	1.717837	0.50000	1.698970

PRODUCTS OF COMBUSTION.

Gas.	CO_2.		H_2O.	
	K.	Log.	K.	Log.
1 K. C to CO_2	3.66667	0.564272		
1 K. CO	1.57143	0.196295		
1 K. CH_4	2.75000	0.439333	2.25000	0.352183
1 K. C_2H_4	3.14285	0.497324	1.28571	0.109143
1 K. H			9.00000	0.954243
1 cu. M. CO	1.43605	0.157169		
1 cu. M. CH_4	1.43605	0.157169	1.17495	0.070019
1 cu. M. C_2H_4	2.87210	0.458198	1.17495	0.070019
1 cu. M. H.			0.587475	1.768990

THERMAL CAPACITIES OF GASES PER KILO.

	Constant Pressure.	Constant Volume.
O	$.213\ T + .000019\ T^2$	$.150\ T + .000019\ T^2$
N & CO	$.243\ T + .000021\ T^2$	$.171\ T + .000021\ T^2$
CO_2	$.193\ T + .000084\ T^2$	$.150\ T + .000084\ T^2$
CH_4	$.608\ T + .000374\ T^2$	$.491\ T + .000374\ T^2$
H	$3.400\ T + .000300\ T^2$	$2.400\ T + .000300\ T^2$
H_2O	$.447\ T + .000162\ T^2$	$.335\ T + .000162\ T^2$
Air	$.236\ T + .00002054\ T^2$	$.166\ T + .00002054\ T^2$

AVAILABLE CALORIFS.

	Per Kilo.		Per Cu. M.	
	Cals.	Log.	Cals.	Log.
C to CO	2,602	3.415307		
C to CO_2	8,080	3.907411		
CO	2,436	3.386677	2,226.14	3.347551
CH_4 (v)	12,200	4.086360	6,370.84	3.804196
" (l)	13,063	4.116043	6,821.50	3.833879
C_2H_4 (v)	11,727	4.069187	10,716.72	4.030061
" (l)	12,142	4.084290	11,095.97	4.045164
H (v)	29,100	4.463893	1,899.50	3.278640
" (l)	34,500	4.537819	2,251.99	3.352565

(v) Water as vapor. (l) Water as liquid.

THERMAL CAPACITIES OF GASES, CALS. PER KILO.

Temp.	Air.		Oxygen.		N and CO.		Hydrogen.	
	Cals.	Log.	Cals.	Log.	Cals.	Log.	Cals.	Log.
100	23.82	1.376942	21.49	1.332236	24.51	1.389343	343	2.535294
200	48.05	1.681693	43.36	1.637089	49.44	1.694078	692	2.840106
300	72.69	1.861475	65.61	1.816970	74.79	1.873844	1,047	3.019947
400	97.74	1.990072	88.24	1.945665	100.56	2.002425	1,408	3.148603
500	123.20	2.090611	111.25	2.046300	126.75	2.102948	1,775	3.249198
600	149.07	2.173390	134.64	2.129174	153.36	2.185712	2,148	3.332034
700	175.35	2.243906	158.41	2.199783	180.39	2.256212	2,527	3.402605
800	202.04	2.305437	182.56	2.261406	207.84	2.317729	2,912	3.464191
900	229.14	2.360101	207.09	2.316159	235.71	2.372378	3,303	3.518909
1,000	256.65	2.409341	232.00	2.365488	264.00	2.421604	3,700	3.568202
1,100	284.57	2.454189	257.29	2.410423	292.71	2.466438	4,103	3.613102
1,200	312.90	2.495406	282.96	2.451725	321.84	2.507640	4,512	3.654369
1,300	341.64	2.533569	309.01	2.489973	351.39	2.545789	4,927	3.692583
1,400	370.79	2.569128	335.44	2.525615	381.36	2.581335	5,348	3.728191
1,500	400.35	2.602440	362.25	2.559008	411.75	2.614634	5,775	3.761552
1,600	430.32	2.633792	389.44	2.590441	442.56	2.645972	6,208	3.792952
1,700	460.70	2.663418	417.01	2.620146	473.79	2.675586	6,647	3.822626
1,800	491.49	2.691515	444.96	2.648321	505.44	2.703670	7,092	3.850769
1,900	522.69	2.718244	473.29	2.675127	537.51	2.730387	7,543	3.877544
2,000	554.30	2.743745	502.00	2.700704	570.00	2.755875	8,000	3.903090
2,100	586.32	2.768135	531.09	2.725168	602.91	2.780252	8,463	3.927524
2,200	618.75	2.791515	560.56	2.748622	636.24	2.803621	8,932	3.950949
2,300	651.59	2.813974	590.41	2.771154	669.99	2.826068	9,407	3.973451
2,400	684.84	2.835589	620.64	2.792840	704.16	1.847671	9,888	3.995108

THERMAL CAPACITY OF GASES, CALS. PER KILO.

Temp.	Carbon Dioxide.		Methane.		Water Vapor.	
	Cals.	Log.	Cals.	Log.	Cals.	Log.
100	20.14	1.304059	64.54	1.809829	46.32	1.665769
200	41.96	1.622835	136.56	2.135324	95.88	1.981728
300	65.46	1.815976	216.06	2.334574	148.68	2.172253
400	90.64	1.957320	303.04	2.481500	204.72	2.311160
500	117.50	2.070038	397.50	2.599337	264.00	2.421604
600	146.04	2.164472	499.44	2.698483	326.52	2.513910
700	176.26	2.246154	608.86	2.784517	392.28	2.593596
800	208.16	2.318397	725.76	2.860793	461.28	2.663965
900	241.74	2.383349	850.14	2.929490	533.52	2.727151
1,000	277.00	2.442480	982.00	2.992111	609.00	2.784617
1,100	313.94	2.496847	1121.34	3.049737	687.72	2.837412
1,200	352.56	2.547233	1268.16	3.103174	769.68	2.886310
1,300	392.86	2.594238	1422.46	3.153040	854.88	2.931905
1,400	434.84	2.638329	1584.24	3.199821	943.32	2.974659
1,500	478.50	2.679882	1753.50	3.243906	1035.00	3.014940
1,600	523.84	2.719199	1930.24	3.285610	1129.92	3.053047
1,700	570.86	2.756530	2114.46	3.325199	1228.08	3.089227
1,800	619.56	2.792083	2306.16	3.362889	1329.48	3.123681
1,900	669.94	2.826036	2505.34	3.398867	1434.12	3.156585
2,000	732.00	2.864511	2712.00	3.433290	1542.00	3.188084
2,100	775.74	2.889716	2926.14	3.466295	1653.12	3.218304
2,200	831.16	2.919685	3147.76	3.498001	1767.48	3.247354
2,300	888.26	2.948540	3376.86	3.528513	1885.08	3.275329
2,400	947.04	2.976368	3613.44	3.557921	2005.92	3.302313

STEAM PRESSURES AND TEMPERATURES.

Absolute Pressure, Lbs.	Gauge Pressure, Lbs.	Temperature, C°.	Absolute Pressure, Lbs.	Gauge Pressure, Lbs.	Temperature, C°.
14.7	0.0	100.0	70.0	55.3	150.2
15.0	0.3	100.6	75.0	60.3	153.0
20.0	5.3	108.8	80.0	65.3	155.4
25.0	10.3	115.6	85.0	70.3	157.8
30.0	15.3	121.2	90.0	75.3	160.0
35.0	20.3	126.2	65.0	80.3	162.2
40.0	25.3	130.6	100.0	85.3	164.2
45.0	30.3	134.5	105.0	90.3	166.2
50.0	35.3	138.3	110.0	95.3	168.1
55.0	40.3	141.6	115.0	100.3	169.9
60.0	45.3	144.7	120.0	105.3	171.7
65.0	50.3	147.7	125.0	110.3	173.4

CONVERSION FACTORS.

		Factor.	Log.	
Cubic meters	x	35.314500	1.547953	= cubic feet.
Cubic feet	x	0.028317	2.452047	= cubic meters.
Kilos.	x	2.204620	0.343334	= pounds av.
Pounds av.	x	0.453592	1.656666	= kilos.
Calories	x	3.968320	0.598606	= B. T. U.
B. T. U.	x	0.251996	1.401394	= calories.
K. per cu. m.	x	0.062428	$\bar{2}$.795378	= lbs. per cu. ft.
Lbs. per cu. ft.	x	16.018400	1.204622	= K. per cu. m.
Cals. per cu. m.	x	0.112371	$\bar{1}$.050652	= B. T. U. per cu. ft.
B. T. U. per cu. ft.	x	8.899110	0.949348	= cals. per cu. m.
Cals. per kilo.	x	1.800001	0.255274	= B. T. U. per lb.
B. T. U. per lb.	x	0.555554	$\bar{1}$.744726	= cals. per kilo.

ABSTRACTS — ANALYTICAL CHEMISTRY.

By E. WALLER.

Foaming on Boiling or Distilling. Fauto (*Zts. angew. Chem.*, XX., 1233). Directing a blast of air or inert gas against the surface of the liquid will stop foaming without diminishing the rapidity of the distillation. The amount of blast required depends upon the violence of the boiling, as well as on the tendency to foam. The speed of distillation is much increased by the possibility of increasing the amount of heat supplied, but the stronger the blast, the greater the tendency to blow atomized liquid into the condenser, a difficulty which can however usually be readily overcome by devices adapted to the conditions of the case in hand.

Alkalinity of Ash Containing Phosphates. Farnsteiner (through *Analyst*, XXXII., 181). Boil 0.2 to 0.3 gm. of the ash for about 3 minutes in a flask with 20 c.c. N/2 HCl and 40 c.c. H_2O. When cold, add 10 c.c. of a neutral 5 per cent. solution of $CaCl_2$ and then 20 c.c. of N/2 NH_4OH. Make the volume up to 100 c.c. with CO_2 free water, mix well, and allow to stand over night. Then pipette off 50 c.c. of the clear supernatant solution, and titrate with N/10 HCl.

Titration of Ammonia. Ronchèse (*Bull. Soc. Chim.* [4], I., 900). The reaction between ammonium salts and formaldehyde is of this type :

$$6HCOH + 4NH_4Cl = 4HCl + (CH_2)_6N_4 + 6H_2O.$$

All of the acid originally combined with the ammonium is rendered free. The process therefore consists in treating the neutral solution of the ammonium salt (which should contain no CO_2 and should be somewhat dilute) with a considerable excess of formaldehyde of about 20 per cent., adding phenolphthaleine indicator, and titrating with decinormal NaOH. Each c.c. = 0.0017 NH_3. Phenolphthaleine is practically indispensable as the indicator in this test, the hexamethylen amine formed, having an action of its own upon such indicators as methyl orange or cochineal. Litmus does not give a sharp change point. If free acid is present it must first be neutralized by NaOH.

Lime and Magnesia in Waters. Burgess (*Analyst*, XXXII., 208). CaO is estimated by precipitating with a known amount of $(NH_4)_2C_2O_4$ and titrating the excess; MgO by precipitation with NaOH and titrating the excess of alkalinity. CaO test requires: (*a*) One liter containing 4 gms. $(NH_4)_2C_2O_4$, 8 gms. NH_4Cl and 4 c.c. NH_4OH (Gr. 0.88); (*b*) permanganate 1.28 gms. per liter (1 c.c. = 0.001 gm. CaO); (*c*) dilute H_2SO_4. (1 : 5 by vol.)

Place 200 c.c. of the water in a bottle of 350 c.c. capacity, add 50 c.c. of the oxalate reagent, mix well and let stand over night to settle.

Treat a blank of 200 c.c. distilled water in the same manner. Next morning syphon off 125 c.c. of the clear liquid, add 5 c.c. of the diluted H_2SO_4, heat and titrate with the standard $K_2Mn_2O_8$.

For the Mg determination the following solutions are required: (*a*) NaOH N/5 and N/10 solutions; (*b*) H_2SO_4 N/10 solution; (*c*) methyl orange 0.5 gm. per liter.

Take 200 c.c. of the water, neutralize carefully with N/10 acid, and boil 10 minutes to expel all CO_2. Add 20 c.c. N/5 NaOH, and make up to 250 c.c. when cooled. Allow to stand over night to settle; syphon off 125 c.c. of the clear liquid, and titrate back with N/10 acid using 0.1 c.c. of methyl orange indicator. A blank is used also in this case, to give a correction, should such be needed.

The amounts prescribed for use are suitable for waters containing not over 30 parts CaO or MgO per 100,000. With harder waters, larger amounts may be necessary, or the sample may be diluted with known proportions of distilled water.

Alkaline Earths, Separation. Caron and Raquet (*Repert. Pharm.* [3], XIX., 64). Acidifying with acetic acid, and addition of chromate precipitates out Ba. In the filtrate, neutralizing, and adding alcohol separates $SrCrO_4$. In the filtrate from that, rendering ammoniacal, and adding ferrocyanide, with possibly more alcohol, separates Ca as calcium potassium ferrocyanide.

Caustic Lime. Hendrick (*Analyst*, XXXII., 320). The method tried was extraction with a sugar solution. The sugar solution should contain at least 10 per cent. of sugar, and the lime should be shaken with it for full four hours. A little alcohol must be added, to prevent formation of sucrate. *Immediate* shaking after adding the reagents is necessary. The proportions used are 5 gms. of sample with 10 c.c. of alcohol, and 10 per cent. sugar solution to make 500 c.c. $Mg(OH)_2$ is practically insoluble in the sugar solution, but some forms of $MgCO_3$ are soluble.

It is noted that CaO by gravimetric analysis in many commercial samples is higher than CaO by direct titration or by the sugar method. The reason is found to be the existence of some silicate. With low grade limes the difference may be as much as 5 or 10 per cent.

Magnesia in Limestones — Field Test. Catlett (*Am. Inst. Min. Eng. Bimonthly Bull.*, 947). The usual limit in most specifications is 5 per cent. MgO in the finished cement, though some make it as low as 3 per cent. These correspond roughly to 3.2 and 1.9 per cent. respectively in the raw limestone. In the field only an approximate estimation can be expected. This is based on the solubility of $Ca(OH)_2$ in sugar solution whereas $Mg(OH)_2$ is insoluble and the density of the precipitate, especially when compared with that from limestones of known composition serves as a guide. The process requires HCl (1 : 1 by vol.); 30 per cent. KOH (free from carbonate); a cold saturated solution of cane sugar; some C. P. $CaCO_3$; standard limestones for comparison; mortar; 40-mesh sieve; measuring spoon to hold approximately 0.5 gm. of 40-mesh limestone; test-tubes marked at 10 c.c., 3 pipettes for the HCl, KOH and sugar solutions respectively; funnels and papers; test-tube rack, lamp, etc.

To conduct the test crush in the mortar; pass through the sieve; measure off in the spoon; transfer to a test-tube and add 1.75 c.c. of the HCl; as soon as may be, boil to expel CO_2; then add enough pure $CaCO_3$ to neutralize the excess of HCl, and dilute to 10 c.c. In another test-tube mix 1.5 c.c. of sugar solution with 1 c.c. of KOH, and dilute to 10 c.c. Then filter the solution of the limestone into this solution — mix the two by shaking, and note the degree of clouding due to $Mg(OH)_2$.

Permanganate Titration of Iron. Pattison Muir (*C. N.*, XCVII., 50). When reducing the Fe by means of Zn, none of the latter metal can usually be allowed to remain in the solution when proceeding to titration. It has been found that the interfering action of the metallic Zn may be stopped by addition of $HgCl_2$ and cooling the liquid. Metallic Hg is precipitated in quantity upon the Zn, and the action is prevented.

Small Amounts of Iron in Copper Alloys. Gregory (*Proc. Chem. Soc. (Lond.)*, XXIII., 306). The salicylic acid reaction is used, the color imparted by the Cu being destroyed by cautious addition of dilute KCN. Sodium acetate must be present. The salicylic acid used in the test is dissolved in acetic acid. Under these conditions the coloration obtained is a deep red, which can be used colorimetrically. Zn or Sb do not interfere. Pb must be removed by H_2SO_4 before applying the test.

Alumina in Silicates. Hinrichsen (*Ber.*, XL., 1497). On decomposing siliceous minerals with HF and H_2SO_4 and subsequently conducting the analysis on the silica-free residue, a deficiency in Al_2O_3 was noted, which was found to be due to a partial volatilization of the Al as F compound. Some F also was retained in the material.

Manganese in Small Amounts. Tanegi (*Gaz. chim. Ital.*, XXXVI., 332). When less than 0.1 per cent. a colorimetric method is preferable. The author has devised a method depending on the solubility of $Mn(OH)_2$ in glycerin. To 5 c.c. of the Mn solution add 3 c.c. of glycerin and then 1 c.c. of a 50 per cent. solution of NaOH. Then add 10 c.c. of a hypochlorite solution containing 3 per cent. available Cl. Permanganate forms at once, and a colorimetric estimation is readily made.

Volumetric for Nickel. Cantoni and Rosenstein (*Bull. Soc. Chim.* [4], I., 1163). Standard solutions of either ferro- or ferricyanide may be used, the end reaction being determined by spot tests (with $FeCl_3$ or $UO(C_2H_3O_2)_2$ in the case of ferrocyanide, or by $FeSO_4$ in the case of ferricyanide). A piece of filter paper is folded, and a drop of the solution on the upper layer is allowed to penetrate to the layer beneath, when the spot test may be made without interference of suspended precipitate. An examination of the capabilities of these methods, and possible influences of free acetic, alkaline salts, etc., indicated that the titrations should be conducted in a solution only slightly acid with acetic. Of course any metals other than Ni should be absent. Sodium salts have the least influence on the results. Ammonium salts have much more effect. The ferricyanide titration is much less influenced by excess of acetic or of salts than that with ferrocyanide.

Nickel in Ores. Pozzi-Escot (*C. Rend.*, CXLV., 1334). Even if Co, Fe or Mn may be present, the method is asserted to be efficient in separating Ni. In a solution of the ore alkaline earth metals are first separated by use of $(NH_4)_2SO_4$ solution. The filtrate is then concentrated, and to it is added a large excess of $(NH_4)_2MoO_4$ in saturated solution, and also some NH_4Cl. The mixture is heated to 80 or 90° C. for some minutes, then stirred frequently for an hour until cold, the cooling being preferably effected by use of ice. All of the Ni is thus precipitated as nickel ammonium molybdate together with most of the iron. Filter, wash with NH_4Cl (sat. sol.) and separate Fe by NH_4Cl with NH_4OH. In the filtrate the Ni can be separated for weighing by any one of many known processes.

Grossmann and Schuck (*Bull. Soc. Chim.* [4], III., 14) assert that the reaction (which is not new) does not cleanly separate Ni from Co, some insoluble compounds of Co being always formed. Hence it is inapplicable as a test for Ni in Co salts, or in presence of that element.

Volumetric for Lead. Bollenbach (*Zts. f. arcal. Chem.*, XLVI., 582). To the Pb solution is added NaOH in amount sufficient to redissolve the precipitate which at first forms. The mixture is then diluted to a known volume, and 50 c.c. of this are run slowly into and excess of standard $K_2Mn_2O_8$ of known volume, which has been mixed with 400 c.c. of hot water, and 5 c.c. of an 8 per cent. NaOH solution. Agitation while mixing the solutions is necessary. The excess of $K_2Mn_2O_8$ is then determined by the use of N/10 $Pb(NO_3)_2$ solution, the end reaction being the disappearance of the pink color, in the supernatant liquid. The reaction occurring is:

$$6Pb(NO_3)_2 + 2KMnO_4 + 12NaOH = 2MnO_2 + 3Pb_2O_3 + 2KOH + 12NaNO_3 + 5H_2O.$$

Red Lead. Partheil (*Arch. Pharm.*, CCXLV., 519). To 2.5 gms. of the sample, add 10 c.c. H_2O, 5 c.c. of lactic acid and 10 c.c. of 25 per cent. HNO_3. A violent reaction ensues; after that is over the insoluble residue is filtered off and weighed. When $H_2C_2O_4$ is used, some PbC_2O_4 may remain insoluble.

Detection of Arsine in the Air. Hebert and Heim (*Bull. Soc. Chim.* [4], I., 573). The reaction with $HgCl_2$ when a paper wetted with such solution is exposed to AsH_3 is well known, but various other gases which possibly might be present may give a yellow tint which simulates that produced by AsH_3. It has been noted that passing the air through a solution of Cu_2Cl_2 will remove those interfering substances, and that if the air so purified is then passed through a U tube containing a slip of paper moistened with $HgCl_2$, the paper will not show a yellow tint unless AsH_3 is present.

Antimony Electrolytically. Dormaar (*Zts. anorg. Chem.*, LIII., 349). Sb deposited from a sulpho salt solution containing KCN was found to be always too high by 2 to 3 per cent. The deposit was found to contain both combined S and O in small amounts. The oxide was formed at the expense of the H_2O of the solution, and not from the air.

Alloys Containing Tin and Antimony. Berg (*Bull. Soc. Chim.* [4], I., 905). The alloy in chips or filings is mixed with 6 to 8 times its weight of a mixture of equal parts of Na_2CO_3 and S in a porcelain crucible. Heat moderately at first to expel the excess of S, then raise the heat. After cooling dissolve in water, which takes up the sulpho salts of Sn, Sb and As, leaving the others. The intensity and period of heating depends on the character of the alloy. Sn yields very readily, and the temperature need not be above dull red. Sb converts to the sulpho salts with more difficulty and requires a full red heat and a longer time. Cd is difficultly and incompletely converted to CdS. Zn in a finely divided alloy yields ZnS with comparative ease, but Fe, Ni, etc., are imperfectly attacked.

Pb when left as PbS by this treatment can be determined by igniting with $(NH_4)_2SO_4$, which converts it to $PbSO_4$.

Mercuric Chloride in Explosives. Hargraves and Rowe (*J. S. C. I.*, XXVI., 813). Very minute amounts of $HgCl_2$ (1 in 200,000) in explosives will cause decided errors in the Abel test. A very close test is therefore necessary. The method depends upon the extraordinary volatility of $HgCl_2$ at comparatively low temperatures. 100 gms. of the explosive is mixed with the same weight of talc and the mixture placed in an Erlenmeyer flask connected to a small flask containing 50 c.c. of water acidulated with 0.75 c.c. H_2SO_4, the escape tube dipping into the liquid. The Erlenmeyer is heated to 100° for two hours, at the end of which time the $HgCl_2$ will have passed over into the solution. Electrolyzing with a strip of gold foil as a cathode will then fix the Hg. If the amount is too small to show any change of appearance on the surface of the foil, the latter may be inserted in a tube, and then heated so as to volatilize the Hg which may be condensed on a small surface of glass.

Mercury in its Ores. Muller (*Bull. Soc. Chim.* [4], I., 1169). When Au and Pb (one or both) are absent, treat an amount of the pulverized ore which is estimated to contain about 1 gm. Hg, with aqua regia in a covered capsule at a gentle heat — after decomposition, evaporate to dryness at about 50° C. Take up with hot water, add about 50 c.c. of a 20 per cent. solution of KI and then Na_2CO_3 until it produces no more precipitate. Add a little NaOH to obtain positive alkalinity. Filter, wash with hot water, and extract the insoluble portion with more KI and Na_2CO_3. The extract and washings should come to about 100 c.c. and will contain all the Hg. The test is made on aliquot portions of this, using for 0.1 gm. Hg 20 c.c. of a 20 per cent. NaOH solution, and the same amount of formaldehyde (35 per cent.) in stoppered flasks for about 20 hours at 25° C. Metallic Hg is precipitated, which may be washed by decantation with alcohol, and then N/10 I in excess (known amounts) is added, and the flasks set aside in the dark until the Hg has been dissolved. If it has collected in large globules, the cautious application of heat may prove necessary to effect solution. The excess of I is then determined by means of standard $Na_2S_2O_3$.

Titanium — Separation from Zirconium. Dittrich and Freund (*Zts. anorg. Chem.*, LVI., 344). A neutral solution of the nitrates is added, drop by drop to a boiling solution of ammonium salicylate (about 10 gms. in 50 c.c.). After boiling for some time, the solution is filtered as hot as possible, and the precipitate — which consists only of basic Zr salicylate is washed with boiling solution of ammonium salicylate. Under these conditions the Ti basic salicylate remains in solution. Thorium may be separated from Ti by the same procedure.

Thorium, Titanium and Zirconium from Iron. Dittrich and Freund (*Zts. Anorg. Chem.*, LVI., 348). If the Fe is completely kept in the ferrous condition, by the basic acetate process, the other elements can be precipitated free from Fe. The solution is treated with H_2S to reduce Fe to ferrous form, neutralized, diluted to 500 or 600 c.c. and then boiled for one or two hours in an atmosphere of CO_2. If rapidly filtered and washed, the precipitate should contain no Fe.

Separation of Iron from Indium. Mathers (*J. Am. Chem. Soc.*, XXX., 209). Reference is made to the various methods proposed for this difficult separation, with suggestions as to the plans proposed. The author finds that the most satisfactory method is that with nitroso beta naphthol (Ilinski and Knorre, *Ber.*, XXVIII., 2728, and XX., 283). The most of the Fe should be removed by some other method, leaving the proportion present as quite small. Not over 0.05 gm. Fe can be readily handled in using the process. The chloride or sulphate solution is evaporated to 20 or 25 c.c., neutralized with NH_4OH, and then one equal volume of 50 per cent. $HC_2H_3O_2$ added. The Fe is then precipitated by addition of acetic solution (50 per cent.) of nitroso β naphthol. The mixture should stand for several hours, and be cold when filtered. Wash with 50 per cent. acetic, and then with water. A very minute amount of Fe_2O_3 may still be found with the In, and some small amounts of In accompany the Fe. By a second precipitation this latter may be reduced to 1 mgm. or less.

Qualitative for Molybdenum. Bettel (*C. N.*, XCVII., 40). In concentrated solutions the addition of H_2O_2 followed by a little excess of NH_4OH produces a brownish red color. The color is discharged by large excess of NH_4OH or by dilution. A perceptible reaction was observed with 0.001 mg. MoO_3.

Separation of the Yttrium Earths. James (*Chem. News*, XCVII., 61). Fractional crystallization of the bromates is the method recommended. The material in the form of oxalates is mixed into a paste with H_2SO_4 and heated until fumes cease to be evolved. The material is then finely powdered, dissolved in ice cold water, and the resulting solution poured over an excess of $Ba(BrO_3)_2$, the mass being well stirred. After settling, and proving by test on a small portion that enough of the reagent has been added, the solution is filtered, and the filtrate boiled down — preferably in casseroles — until a drop on the end of a rod just solidifies when stirred on a watch glass. It is then cooled and filtered; the mother liquor boiled down again and so on. The order of solubility shown by the earths was: Samarium, terbium, yttrium, dysprosium, holmium, erbium, thalium and ytterbium.

Praeso-dymium and Neo-dymium. Milbauer (*Zts. f. anal. Chem.*, XLVI., 657). In a borax or phosphorus salt bead neodymium affords in the oxidizing flame, colorless; in reducing flame-violet if the element is present in considerable quantity. Præso-dymium gives, in O flame yellowish green, in R flame green. The colorations are only seen when the beads are cold.

Chlorates in Presence of Perchlorates. Venditori (*Gazetta*, XXXVII., ii, 383). On adding H_2SO_4 and powdered Al, chlorates are reduced to chlorides, while perchlorates are unaffected. The chloride thus produced can be estimated by the Volhard method, and the amount of chlorate calculated.

Iodate in Presence of Chlorate. Gibson and Grieb (*Analyst*, XXXII., 381). In a solution containing free acetic acid but no free mineral acid, KI affords free I if iodate is present, but none if only chlorate is present. By this reaction the amount of iodate may be estimated by titrating the free I with standard $Na_2S_2O_3$. An excess of acetic is necessary or the starch blue coloration disappears prematurely.

Small Amounts of Fluorine.. Steiger (*J. Am. Chem. Soc.*, XXX., 219). The colorimetric method described depends upon the bleaching effect produced by F upon the color obtained by adding H_2O_2 to a Ti solution.

A good colorimeter accurate to within about 2 per cent., is necessary. The details must be sought in the original paper.

Titrating Thiosulfate. Gutmann (*Fres. Zts. f. anal. Chem.*, XLVI., 485). In alkaline solution thiosulfate reacts with KCN on heating thus:

$$Na_2S_2O_3 + KCN = Na_2SO_3 + KCNS.$$

The reaction takes place quantitatively and is not affected by the presence of sulfite.

The solution of the substance to be examined is made approximately tenth normal with respect to thiosulfate, and to 50 c.c. of this is added about 1 gm. KCN and 2 c.c. of 15 per cent. NaOH and the mixture is heated for half an hour on the water bath. After cooling it is diluted to 100 c.c. and at least 20 c.c. is taken for a test. The surplus KCN is titrated by Liebig's method with N/10 $AgNO_3$. Then a known excess of N/10 $AgNO_3$ is added and after acidifying this is filtered, the filtrate made up to known volume, and in an aliquot portion the excess of Ag is titrated back (by Volhard's method) with standard KCNS solution.

Nitrogen in Celluloid. Utz (*Gummi Ztg.*, 1907, 103). The celluloid is digested with NaOH solution with addition of H_2O_2, which dissolves all nitro oxides as nitrate and nitrite. Then acidifying and adding more H_2O_2 converts all into nitrate, which may be determined by precipitation with "nitron."

Ozone, Hydrogen Peroxide and Nitrogen Peroxide in gas mixtures. Keiser and McMaster (*Am. Chem. Jour.*, XXXIX., 96). If the mixed gases are drawn through a dilute solution of $K_2Mn_2O_8$ only O_2 will escape

decomposition, and if the gases after that treatment color a blue paper moistened with KI and starch, O_3 is present. Similarly, only NO_2 escapes decomposition by passing through a tube filled with powdered MnO_2 (one 40 cm. in length was used) and may be detected by decolorizing $K_2Mn_2O_8$ solution. Its presence may be confirmed by passing the gases through a solution of nitrite-free NaOH, and then applying the Griess test (sulphanilic and naphthylamine).

To identify H_2O_2 the gases should be passed through a solution containing potassium ferricyanide and ferric chloride. The yellow-brown solution becomes green and then blue, under the influence of H_2O_2, but is unaffected by O_3 or NO_2. The reagent will itself in time give a green precipitate, and needs to be freshly prepared. Also, if the time of passing the gas is much prolonged, a greenish precipitate may form when no H_2O_2 is present.

Sodium Peroxide. Niemeyer (*Chem. Ztg.*, XXXI., 1257). Weigh out 0.15 to 0.2 gm. of the sample in a small glass capsule standing in the bottom of a small weighing glass made from the bottom of a cut-off beaker. Then taking the weighing glass with the forceps plunge the whole in an upright position into a beaker of water (about 500 c.c.). Immediately add excess of H_2SO_4 and titrate with standard permanganate. The NaO_2 should not dissolve at or near the surface, because such conditions may easily permit of loss of O or H_2O_2 to the extent of 5 or 6 per cent.

Detection of Moisture. Biltz (*Ber.*, XL., 2183). $2KI.PbI_2$ is colorless, but is partially dissociated by moisture giving the yellow tinge of PbI_2. The reagent may be prepared thus: A filtered, warm solution of 4 gms. $Pb(NO_3)_2$ in 15 c.c. of water, is mixed with a warm solution of 15 gms. KI in 15 c.c. of water. PbI_2 first precipitates — on cooling the yellow color diminishes, until the mass becomes a network of interlocking white crystals of the double salt. After sucking dry as possible on a filter, this is dissolved in 15 to 20 c.c. of acetone, which is filtered. The reagent can be used in this state, or the salt may be precipitated out by adding double the volume of ether. The aceton solution may be used to impregnate filter paper, which slowly turns yellow from the moisture in the atmosphere (immediately if breathed upon). Such a paper may be used in many ways for the detection of minute amounts of moisture. Moistening with aceton restores the colorless condition.

Carbon Bisulphide in Illuminating Gas. Harding and Doran (*J. Am. Chem. Soc.*, XXIX., 1480). The gas is first passed through KOH and conc H_2SO_4 to remove CO_2 etc., then through a solution of KOH in absolute alcohol which absorbs the CS_2 forming xanthate. After heating the solution, standard $Cu(C_2H_3O_2)_2$ solution is added, the Cu xanthate filtered off and the excess of Cu determined in the filtrate.

Benzene in Illuminating Gas, etc. Dennis and McCarthy (*J. Am. Chem. Soc.*, XXX., 233). The ammoniacal solution of $Ni(NO_3)_2$ recommended in 1903, was found to be efficient in some cases, and not in others. The reason was found to be in the presence of CN and experiments demonstrated that an ammoniacal solution of $Ni(CN)_2$ was

the best reagent, especially after it had been treated with a little benzene. The reagent is prepared by dissolving 50 gms. crystallized ($NiSO_4.7H_2O$) in 75 c.c. of water, adding 25 gms. KCN in 40 c.c. of water, and then 125 c.c. NH_2OH (Gr. 0.91). After thorough shaking the mixture is allowed to stand for 20 minutes at 0°. The clear solution is then decanted, and mixed with a solution of 18 gms. citric acid in 10 c.c. of water. Two drops of benzene (C_6H_6) are then added, and the whole shaken until the benzene has been taken up. The reagent can be used over and over, its effectiveness being increased by repeated use, of course within the limit of the presence of a reasonable amount of the Ni compound. The product of the reaction with benzene is $Ni(CN)_2NH_3C_6H_6$ which separates as precipitate, and in course of time must be removed by filtration.

Absorption of C_6H_6 by means of H_2SO_4 (Gr. 1.84) was found to give irregular results, and moreover the reagent removes indeterminate amounts of ethylene (C_2H_4).

METALLURGICAL ABSTRACTS.

By WILLIAM CAMPBELL.

ANTIMONY.

Antimony Ore Smelting in Australia: E. and M. J., 84, 1109. From *Queensland Government Mining Journal*, Aug., 1907. Works at Balmain, Sydney, of Metals Smelting Co. Reverberatory furnace and plant started smelting in Feb., 1907. 24 tons of star antimony regulus per month. Extraction 91 per cent.

COPPER.

Copper in 1907: the Metallurgy of: W. R. Ingalls. *E. and M. J.*, 85, 25. New plants are Garfield, Chrome, Balaklala in Shasta, Co., Cal. The Imperial at Sasco, Ariz., and the Steptoe at McGill, Nev. Blast furnaces *vs.* Reverberatories. All the new works have reverberatory smelters with a certain number of blast furnaces.

The Copper Belt of California—III.: H. Lang. *E. and M. J.*, 84, 1006. Mentions the early smelters as Fresno, Buchanan, James' Ranch, Victory, Copperopolis, Campo Seco, Newton, Copper Hill and Spencerville mines. The later smelting operations are taken up. Smelting at Copperopolis. Pyritic smelting at the Fresno smelter.

The Smelter of the Mammoth Copper Mining Co. at Kennett, Cal.: D. F. Campbell. *M. and S. P.*, 96, 30. Owned by the U. S. S. R. and M. Co. 3 furnaces 42 × 180 in. with 6 ft. 9 in. ettlers, pan conveyors which. discharge into barrows. Matte shipped to Utah plant. Slag discharges into swivelpot and overflows into slagpot — to dump. 20 to 30 per cent. matte formed.

The Tyee Smelter: R. L. Phelps. *M. and S. P.*, 95, 782. At Ladysmith, on Vancouver Island; 200-ton Allis-Chalmers 42 × 120-in. furnace, 10 ft. column; runs through 250 to 300 tons per 24 hrs.; pressure, 1½–2 lbs.; fore hearth 10 ft. long, 4 ft. deep, has pocket with matte spout, similar to a "siphon" settler. Ore runs 4 per cent. Cu; matte 40 per cent. Shipped to Tacoma. Slag granulated.

Utah's Largest Copper Smelter: R. B. Brinsmade. *Mines and Minerals*, 28, 305. A description of the Garfield Plant. Two classes of ore: crude pyritic, 2 to 4 per cent. Cu; $.50 to $4.00 Au. 1 to 4 oz. Ag: 20 mesh concentrate 20 to 25 per cent. Cu. $.20 Au and Ag: from Bingham. Other ores from Newhouse, Utah, southern Utah and Nevada. Elaborate system of bins, tracks and belt conveyors for handling ores. Two blast furnaces in operation, two more building; 4 ft. × 20 ft. at tuyeres, 16 ft. settlers, slag overflowing into train of 10 ton Nesmit pots. Matte runs 20 per cent. Cu, part of which is mixed with 40 per cent. matte from reverberatories and converted. Rest is cast and rerun on certain days for a 30 per cent. matte, which

is converted; 26 H. and H. pots for copper ores preroasted in McDougals. Three 19 ft. × 112 ft. reverberatories, grate 7¼ ft. × 16 ft. Four converter stands, 8 ft. dia. × 12 ft. long. Elaborate system of flues.

The Steptoe Valley Mill and Smelter: W. R. Ingalls. *E. and M. J.*, 84, 813. Works being built jointly by the Nevada Consolidated and the Cumberland-Ely. Situated on the McGill ranch, 14 miles northeast of Ely. Capacity 4,000 tons crude ore per diem. Concentrating mill. Smelting works, a reverberatory plant, sixteen 18-foot McDougalls — 640 tons per diem to 7.5 per cent. S; 1,500-ton brick and steel storage bin for surplus calcines. Three 19 × 112-ft. reverberatory smelters: each with two 400 h. p. boilers in parallel. Grates are 18 ft. × 19 ft.: 3 ft. high to top of fire bridge. Two charge hoppers for ore, etc. Matte tapped into ladles on cars for the converting department: slag granulated. Flues, etc. (300 sq. ft. section). Three converters 96 × 150 in. Electrically driven. Siliceous Tonopah ore for lining and charging. Flue 100 sq. ft. section. Blister poured from converters into casting machine of conveyor type. Blast furnace 42 × 240 in. × 16 ft. Each jacket independent: has four tuyeres. Expect to run with 10 ft. column and 24 to 30 oz. pressure. Photographs show the plant in the course of construction.

Converter — Matte from Copper-Concentrates by Pot-Roasting and Smelting: The Production of: G. A. Packard. *Bull. A. I. M. E.*, 1908, 75. Describes laboratory experiments in pot-roasting and smelting of jig and table concentrates containing 6.32 Cu, 38.85 per cent. S and 4.65 per cent. Cu, 44.10 per cent. S, respectively. Preliminary tests were made on a "Size N" Battarsea crucible. The large pot was 25 in. in dia., 23 in. high, ⅛ in. sheet iron grate with ⅜ in. holes. Pressure 8–14 in. of water.

Precipitation of Copper from Cupriferous Waters: F. H. Probert. *M. and S. P.*, 96, 27. At Rio Tinto, Huelva, Spain, huge masses of ore are treated by "natural cementation." Massive pyrite containing from a mere trace to 4 per cent. Cu. Heaps are irrigated continuously with water and spent liquors. The copper liquor as it leaves the heaps contains much ferric iron, which is reduced by passing it through a small heap of fines of raw ore. The precipitation vats are a series of canals, wood-lined, containing a chequer work of pig iron, 4 pigs deep. 1.4 units of pig iron are consumed per unit of copper obtained, at Rio Tinto and Tharsis.

At Butte, along Silver Bow Creek, enormous masses of tailing and slag carry copper. The waters of the creek are acid, and could be used upon the tailings. The general method of precipitation at Butte is slipshod.

Gold and Silver.

Gold Ore Treatment during 1907, Progress in: A. James. *E. and M. J.*, 85, 17. Takes up crushing. Slime treatment. Vacuum filters — the Ridgeway. Butters-Cassel *vs.* Moore. Merrill's plant. The Brown agitator, at the Komata Reefs. N. Z. Cyanidation of silver-gold ores. (Also *M. and S. P.*, 96, 41.)

Gold Dredging in 1907: J. P. Hutchins. *E. and M. J.*, 85, 18. Deals with dredging frozen ground, salted ground, search for new areas, misdirected exploitation, prospecting dredging ground, variation in practice, dredging design, steam shovel dredging, difficulties at Oroville, Cal. Costs, etc.

The Great Gold Mines: T. A. Rickard. *M. and S. P.*, 96, 11. The most productive and profitable in 1906 was the Esperanza. In 1907 the most productive was the Goldfield consolidated, the most profitable was the Robinson. A table of the 19 principal mines of the Transvaal is given, another of the leading mines of Australasia. The third district is Cripple Creek, Colo., though it can claim no mine of the first magnitude. Next comes the Kolar dist., India, then the El Oro, Mexico, with the Esperanza, El Oro and Dos Estrellos.

Dredging Beach Gravel Deposits near Nome: J. P. Hutchins. *E. and M. J.*, 84, 955. Describes types of early dredges. Beach deposits. Tundra gravel. Stream gravel. Methods of prospecting. Operations.

Gold Mining in Colombia: J. de la C. Posada. *E. and M. J.*, 84, 827. Quartz mining with Cornish Wooden Mills. Stamps 400 or 500 lbs. Drop 30 to 70 per min. Crush ½ ton per 24 hrs. Gold caught in blankets. Pyrite ground in arrastra. Extraction 50 per cent. $16 ore is worked. General value of tailings about $10. Cyanide would save much of this. One smelter, the Zancudo in Antioquia.

The Roasting of Telluride Ores: R. L. Mack and G. H. Scibibd. *M. and S. P.*, 95, 751, 777. Describe experiments in roasting of tellurides to find the effect of size of crushing, temperature, loss of gold, distribution of gold in various sizes of crushed ore. Ore from Portland mine used. The degree of fineness under 14-mesh does not affect the amount of desulphurization, the important factor being high temperature with a sufficient amount of air. The average percentage loss was 2.04 per cent., being the same for sizes through 150-mesh to 60-mesh inclusive. The object in roasting is not the elimination of sulphur but of tellurium, the elimination of sulphur being an indication that tellurium is also eliminated. Calaverite melts at 365° C. The ore was fritted at 1095° C.

History of Cyanidation: P. Argall. *M. and S. P.*, 95, 655, 685. A paper read before the Colorado Scientific Society, giving the history of the process from the time of Diesbach down to the use of vacuum filters in modern times.

Cyanidation with the Brown Vat: F. Narvaez. *M. and S. P.*, 95, 689. The San Francisco mill near the El Sotol mine was designed for pan amalgamation and transformed into a cyanide plant recently. Mill of thirty 1050 lb. stamps, 90 a minute, through 7 in., using Tyler double-crimped 16-mesh wire cloth. 2.8 tons a head. Wilfly table. 15 per cent, AuAg in concentrates. Conical hydraulic classifier. Overflow to the Pachuca "Agitating Tanks," patented by A. Grothe (invented by Brown of W. A.). Steel cylinders, conical bottoms, 30 ft. + 15 ft. high, 15 ft. dia., in the axis is a 15-inch pipe into the lower end of which a 1½-inch pipe from the compressor line points.

Causes circulation. Underflow from classifier reground in Boss pans. Will be replaced to 2 Krupp tube-mills.

Grinding in 0.13 to 0.17 per cent. KCN solution. Make up to 0.3 per cent. for agitation, 130 gms. acetate of lead being added. Agitate 48 hrs., pulp 1 : 1.47. Butters filter 28 leaves; need to be doubled.

The Burt Rapid Cyanide Filter : E. Burt. *M. and S. P.*, 95, 717. At the El Oro the press was worked out. Is a cylinder pivotally mounted or set in a fixed position at an angle of about 45° ; large door at lower end. It carries a series of filter matts swingingly suspended in its interior. Used with a pressure of 40 to 60 lbs. Drawings illustrate the filter. Data given.

Slime Treatment at Kalgoorlie : M. W. Von Bernewitz. *M. and S. P.*, 95, 743. At the Associated Northern the ore carries tellurides and iron pyrite, very little free gold. No. 5 gates crusher, three No. 5 Krupp ball mills through 27 mesh. 6 Merton furnaces. Mix with weak KCN solution, grind to slime and amalgamate the coarse gold in 8 Forwood-Down 5-ft. pans. Settle slime in V-shaped boxes, agitate pulp in five vats 22 × 6 ft. Filter-press in three Dehne presses. Disposal of residue by belt conveyor. Pan product, 87.5 per cent. through 150 mesh. Costs, agitation and cyaniding, $0.34. Filter-pressing, 0.41. Precipitation, 0.12. Disposal of residues, 0.04. Total, $0.91.

Slime Treatment: A. M. Nicholas. *M. and S. P.*, 95, 583. A letter describing a patent (U. S. Pat. No. 619,211) taken out in Feb., 1899, for a slime filter, with the idea of cleaning the filter by compressed air, etc., thus antedating the Moore patent. (See p. 715.)

Iron and Steel.

Segregation in Ingots : A Further Study of: H. M. Howe. *E. and M. J.*, 84, 1011. Concludes that between the limits of 2¼ in. square and 16 in. square the influence of ingot size on the degree of segregation is of such slight importance that it is masked by that of other variables. Very large ingot size (30 in. across and more) increases segregation. In most cases there is more segregation in ingots cooled quickly than in those cooled slowly. The most enriched spot lies in the axis of the ingot usually at a distance from the top of between 6 and 28 per cent. of the ingot's length. Numerous curves illustrate the paper.

Steel Rail Specifications : E. and M. J., 84, 1212. Adopted by Carnegie Steel Co., Nov. 1, 1907.

Sections.	Range in Per Cent.		Limit in Per Cent.
	Carbon.	Manganese.	
50 to 60 lbs.	0.35 to 0.45	0.70 to 1.0	0.10 Phosphorus.
61 to 70 "	0.35 to 0.45	0.70 to 1.0	0.20 Silicon.
71 to 80 "	0.40 to 0.5	0.75 to 1.05	
81 to 90 "	0.43 to 0.53	0.80 to 1.10	
91 to 100 "	0.45 to 0.55	0.84 to 1.14	

On leaving the rolls at the final pass the temperature of the rails of sections of 75 lbs. per yard and heavier will not exceed that which

requires a shrinkage allowance at the hot saws of $6\frac{7}{16}$ in. for a 33 ft. 75 lb. rail, with an increase of $\frac{1}{16}$ in. for each increase of 5 lbs. in weight of the section.

LEAD.

Lead in 1907 — The Metallurgy of: W. R. Ingalls. *E. and M. J.*, 85, 23. Deals with the Huntington and Heberlein process. At Murray plant 25 pots 9 ft. dia. roast 9 tons of ore each in 12 hrs. 6 men on the floor. 15 to 12 per cent. of H. and H. material used in the blast furnace charge and the capacity of a 48 × 168 in. furnace increased from 175 to 200 tons per diem. At the lead smelter of the U. S. Co., at Bingham Junction, 15 hand roasters take 12 tons of ore or 15 tons matte a day. Cyrus Robinson installed an experimental plant of "blast-roasters" or grates. Fume filtration, etc. Lead smelting in Mo. and Ill. Five pot Savelsberg installation at Herculaneum. H. and H. at the Federal Lead Co. The Scotch Hearth and bag house at Collinsville (St. Louis S. & R. Co.) extract 98½ per cent. of the lead. Lead runs 2–3 oz. Ag, and is refined by the Parkes process to produce corroding lead.

The Lead Silver Mines of Eureka, Nevada: W. R. Ingalls. *E. and M. J.*, 84, 1051. From 1869 to 1879 they were the largest domestic supply of pig lead. At the end of 1870 there were 14 furnaces at Eureka. Ore 40–48 per cent. lead, $60 to $80 in silver, $15 to $20 in gold per ton. Pig lead ran $170 Ag and $80 Au. The Eureka Con. M. Co., the largest producer, the Richmond Con. second. In 1874 the Richmond Co., erected a refinery. In 1890 operations came to a standstill. The paper deals with the geology of the ore bodies, and the recent developments.

Lead Mining in the United States: Chronology of: W. R. Ingalls. *Bull. A. I. M. E.*, 1907, 979. Gives a brief history of lead mining and smelting in the U. S.

History of Smelting in the Joplin District: D. Brittain. *E. and M. J.*, 84, 861. Describes log furnaces used by the Osage Indians and hunters. Air furnaces and hearths. Early smelting operations in the Scotch hearth. Moseley mines, two Scotch hearths 3,000 lbs. ore per 12 hrs. (1852). The second smelter was that of Hartlerhodes in 1855, on Hickory Creek, 5 miles S. E. of Neosho. In 1856 three Scotch hearths erected at Granby, in 1865 reorganized as the Granby Mining and Smelting Co. Eight Scotch hearths, 3,000 lbs. in 7 hrs., succeeded by 5 waterback Scotch hearths, 7,000 lbs. in 7 hrs., and these were followed by five waterback Jumbo Scotch hearths 14,000 lbs. ore in 7 hrs. Method of operation described.

Other pioneer smelters were: At Oronogo, 1859. The Joplin Mining and Smelting Co. in the Kansas City bottoms (the Kansas City Mining and Smelting Co.): three or four air furnaces and 6 or 8 Scotch hearths. 1871 Moffett and Sargeant smelter, lease of the Lone Elm tract; finally became the Picher Lead Co. Bartlett process. 1879 the Galena Lead and Zinc Co. The Petraeus Co. (Galena S. and Mfg. Co.) 1902. Plant described. See *E. and M. J.*, 84, 885, for map of district.

ZINC.

Zinc Smelting in the United States in 1907: W. R. Ingalls. *E. and M. J.*, 85, 20. Takes up the production and general conditions in 1907. Describes the Bartlesville Zinc Co., the National Zinc Co., smelting practice at Iola, etc.

Name.	Location.	Furnaces.	Retorts.
Grasselli Chemical Co.	Clarksburgh, W. Va.	10	5,760
Matthiessen & Hegeler.	Lasalle, Ill.	5	4,320
Illinois Zinc Co.	Peru, Ill.	7	4,800
Sandoval Zinc Co.	Sandoval, Ill.	2	448
Mineral Point Zinc Co.	Depue, Ill.	6	4,800
Hegeler Bros.	Danville, Ill.	2	1,700
Edgar Zinc Co.	St. Louis, Mo.	9	2,000
" " "	Cherryvale, Kan.	24	4,800
Lanyon Zinc Co.	Iola, Kan., 3 plants of	5	3,000 ea.
United Zinc and Chem. Co.	Iola, Kan.	4	2,304
" " " " "	" "	2	(a) 480
" " " " "	Springfield, Ill.	2	640
Cockerill Zinc Co.	Iola, Kan.	5	3,000
" " "	" "	3	1,800
" " "	Altoona, Kan.	5	3,000
" " "	Pittsburg, Kan.	3	672
" " "	Nevada, Mo.	2	448
" " "	Rich Hill, Mo.	2	448
Granby M. & S. Co.	Neodesha, Kan.	6	3,840
U. S. Zinc Co.	Pueblo, Colo.	6	(a) 1,440
American Zn, Lead & S. Co.	Caney, Kan.	6	3,720
" " "	Deering, Kan.	6	3,720
N. J. Zinc Co.	Bethlehem, Pa.	3	672
" " "	Palmerton, Pa.	12	2,400
Bertha Mineral Co.	Pulaski, Va.	10	1,400
Prime W. Spelter Co.	Iola, Kan.	9	5,344
" " "	" "	5	3,229
Bartlesville Zinc Co.	Bartlesville, Okla.	6	3,456
Pittsburg Zinc Co.	Pittsburg, Kan.	3	672
Lanyon Starr S. Co.	Bartlesville, Okla.	6	3,720
National Zinc Co.	" "	4	2,432
Chanute Zinc Co.	Chanute, Kan.	8	1,600

(a) Rhenish furnaces with muffles.

Electric Zinc Smelting: F. T. Snyder. *M. and S. P.*, 96, 720, abs. Trans. Tristate Mining Association. Compares the cost of the usual retort method with that of the electric stack method and figures that the latter costs $2.50 less per ton of raw ore.

PYROMETRY, METALLOGRAPHY.

Solidification and Transformation in Alloys: M. A. Portevin. *Revue de Métallurgie*, 1907, 915. The application of the phase rule to the equilibrium of a binary system.

Constitution of Alloys from their Electromotive Force: M. Pouchine. *Rev. de Mét.*, 1907, 926. Describes methods employed and results obtained. The following compounds are indicated from the curves: Ag_2Se, Ag_2Te, Cu_2Te and CuTe, PbTe, SnTe, Zn_6Cu, Zn_2Cu, ZnCu, $ZnCu_2$. The same compounds for Ag and Zn, together with Zn_4Ag, $AuZn_6$, $AuZn_2$ and AuZn, Cd_2Cu, $SnCu_2$ and $SnCu_8$, $SnAg_2$, $AuSn_2$,

AlCu and AlAg. Antimony and bismuth give a series of isomorphous mixtures: Zn_3Sb_2 and ZnSb, Sn_3As_2 and SnAs, SbSn, $SbAg_2$ and $SbAg_3$, SbNi and $SbNi_2$, SnCo and SnNi, $SnMn_3$ and Fe_4Sn.

The Alloys of Cobalt and Copper: N. Konstantinow. *Rev. de Mét.*, 1907, 983. Fixed points taken were cobalt, 1505° C.; nickel, 1484°; copper, 1084°. The curve consists of two branches which intersect at 6.5 per cent. Co at 1105° C., while the upper branch is cut by the critical curve forming two conjugate liquid solutions of 30 and 70 per cent. Co.

Iron-Carbon System: Present State of the Theories of the Equilibrium of: M. A. Portevin. *Rev. de Mét.*, 1907, 993. Shows the diagrams of the two systems, stable with graphite, labile with cementite, and discusses the conditions of the passage of one system into the other.

The Alloys of Copper and Bismuth: M. A. Portevin. *Revue de Mét.*, 1907, 1077. Gives the curve which falls slowly from pure copper to 95 per cent. Bi at 800° C. and then a sharp fall to the freezing point of bismuth. The alloys consist of grains and dendrites of copper set in a groundmass of bismuth.

GENERAL.

Metallurgical Calculations: J. W. Richards. *Electro-Chem. and Met. Ind.*, V., 496. Electric smelting of copper ores. Works out the balance sheet, etc., for some of Vattier's experiments at the Livet works of the Compagnie Electrothermique Keller et Leleux.

Smelter Smoke, with a Discussion of Methods for Lessening its Injurious Effects: L. S. Austin. *M. and S. P.*, 95, 649. Describes the formation of flue dust and fume. The amount of SO_3 in lead smelting is not more than 0.2 to 0.5 per cent., in copper smelting 0.9 to 2.5 per cent. In the copper reverberatory at the Washoe plant percentage formed was 2.08 during the first 10 minutes and drops to 1 per cent. during the succeeding 80 minutes of melting. The action on vegetation. Remedies proposed such as H_2SO_4 mfr.: Operating in a barren country: Roasting in a barren country: Filtering all fume, etc.

By-product Coke Oven in the United States: C. G. Atwater. *Mines and Minerals*, 28, 317. Gives table and information of by-product coke ovens built or building in the U. S. at end of 1906. Location, number of ovens, use of coke and use of surplus gas.

Variations in Mining Costs: J. R. Finlay. *M. and S. P.*, 96, 23. Gives comparative table of working costs for the Alaska-Treadwell, Esperanza, El Oro, Cripple Creek Dist., the Portland and the Camp Bird. For the Anaconda, Boston and Montana, Calumet and Hecla, Wolverine, Quincy, Mohawk, Tennessee. For the Bunker Hill and Sullivan, Federal M. & S. Co., Idaho, and S. E. Missouri.

Mining in Australia: H. L. Wilkinson. *M. and S. P.*, 95, 616. Gives the output of gold, copper, silver, lead, etc., for N. S. W., West. Australia, Queensland and Victoria. Tables of figures for the twelve great gold mines of Western Australia; for the mining districts of Victoria and Queensland. There are 172 dredging plants at work in New Zealand. In vein mining the Waihi and Talisman Considated contributed the greater part of the gold output.

BOOK REVIEWS.

Mining, Mineral and Geological Law. A Treatise on the Law of the United States, Involving Geology, Mineralogy and allied Sciences, as applied in Mining, Real Estate, Public Land, United States Customs and other Litigation. Also the Acquisition and Maintenance of Mining Rights in the Public Domain and obtaining Patents for Mineral Land under the United States Mining Laws. By CHARLES H. SHAMEL, M.S., LL.B., A.M., Ph.D. Hill Publishing Company, New York, 1907. 8vo, pp. xxx + 627, 101 illustrations. Price $5.00.

Under this somewhat formidable title is issued a treatise which has for its primary object a concise presentation of the legal principles governing mining rights upon the public lands of the United States. The book is intended for the use of those interested in the mining industry as well as for the legal fraternity. To this end many purely technical legal expressions are either avoided, or explained in sufficient detail to be readily understood by the layman. At the same time an effort has been made to deal with the scientific matter concerning geology and ore deposits in such a manner as to give little difficulty to those who have not undergone a technical training.

The scope of the work may best be indicated by reference to the table of contents. Chapter I., Introductory: Sources of American Mining Law; territorial divisions within which the various laws apply. Chapter II., Geology and allied sciences outlined and defined. Chapter III., Property in minerals; historical sketch of the law; minerals under highways and navigable waters; oil and natural gas; mill tailings, etc. Chapter IV., Legal definitions of mineral and ore. Chapter V., Theories of ore formation; views held by modern geologists; classification of ore deposits. Chapter VI., Right of extralateral pursuit of vein; origin of the principle. Chapter VII., Public Domain and Mining Laws applying thereto; qualifications of locators. Chapter VIII., Acquisition of mining rights; actual discovery essential; quantity of mineral necessary; definition of term "In place." Chapter IX., Location; marking of boundaries; notices; assessment work; forfeiture; relocation; patenting. Chapter X., Scientific definitions of various kinds of veins. Chapter XI., Legal definitions of vein or lode; the leading cases; classification of the same. Chapter XII., Legal definitions of apex, strike, dip, etc.; right of inspection where underground trespass is suspected. Chapter XIII., Extralateral rights; position of lode with respect to boundaries of the claim; rules derived from adjudicated cases. Chapter XIV., Extralateral rights continued; form of location; rights to "secondary veins," apexing within a location; cross veins; horizontal veins; conflict with agricultural patent. Chapter XV., Tunnel locations; exception to apex law. Chapter XVI., Discussion of the proposal to repeal the extralateral right. Chapter

XVII., Placers and placer locations. Chapter XVIII., Waters: Surface and subterranean water; tide lands; riparian rights; pollution of underground waters. Chapters XIX. and XX., Miscellaneous uses of geology in law; obliterated boundaries; accretion; lateral and sub-adjacent support, etc.

An appendix contains the federal statutes in full, the latest revision of state and territorial laws, and the Land Office rules and regulations.

The bibliography, included in pp. 551 to 570, furnishes a list of the latest and best publications on geology, mineralogy, mining, metallurgy and mining law, which will be appreciated alike by those engaged in mining and those whose legal practice brings them in contact with these subjects.

Throughout the text numerous illustrations are introduced. Many of these are diagrams of locations which have given rise to litigation, while others are for the purpose of explaining geological conditions. Perhaps among the most noteworthy features of the book may be mentioned the attention devoted to geology, the modern theories of ore formation, the scientific distinctions with respect to various kinds of deposits, and the classification of the numerous varieties of rocks encountered in mining operations. The work indicates a great amount of careful research on the part of the author who has been particularly successful in presenting the latest scientific data and theories, together with the most recent legal decisions. Citations of the most important cases bearing upon the subjects under discussion are inserted as footnotes, and reference to the whole work is greatly facilitated by the comprehensive index.

In discussing the proposal to repeal the extralateral right the author endeavors rather ingeniously to defend and justify this feature of the law, as follows (p. 261):

"It is inconceivable that in all these different countries the rule of allowing the vein to be followed on the dip without reference to surface boundaries should have been developed and exercised, if it was unjust or not, the best possible rule for existing conditions. In the case of California, the re-enactment of the same rules in hundreds of independent mining camps, by men absolutely untrammeled by any law or custom except their innate sense of justice and fair play proves that it is simply impossible that it should have been otherwise."

Unfortunately this argument attempts to prove too much to be regarded as a very stable support for our extralateral law. Referring to page 235, concerning the litigation that arose with respect to the flat deposits of the Leadville district, the author says:

"There was a very strong local feeling in the district against allowing extralateral rights to the fortunate claims which were located on the outcrop of the vein. . . . For although the law might be correctly stated in the instructions, the jury would nevertheless decide that the ore-body in dispute in the particular case was not a vein or lode, and consequently deprived it of extralateral rights." Were not such jurors equally "untrammelled by any law or custom except their innate sense of justice and fair play"?

In the early days of mining in the west the "dip right" was adopted

by the miner because he acquired thereby, in a very direct way, that which he valued most, the right to the lode, entirely independent of any surface area that he might locate for the convenient working of his mine. But ultimately, when litigation arose concerning similar locations, the courts evidently were not impressed with the idea that this was "the best possible way," for they read into the law that certain surface boundaries were implied, and proceeded to determine the miner's right to the lode in accordance with the position of such surface boundaries.

Both Germany and British Columbia experimented with the extralateral right and finally abolished this feature of their laws on account of the frequent and serious litigation to which it gave rise. To-day the extralateral right appears to exist only in the United States, and even in portions of this country (Arizona) we find mining companies contracting with one another to waive the extralateral feature and abide by the limitations fixed by vertical planes passing through the surface boundaries.

Among the best reasons presented by the author for the retention of the extralateral right in our mining statutes is the following (p. 271):

"But, as heretofore mentioned, there is another very weighty reason against substituting another rule for that of extralateral rights, in the fact that the latter has been the law for over forty years, during which time thousands and thousands of locations have been made and many of these developed into properties of enormous value. Existing extralateral rights could not be abrogated by the enactment of a new system, but must continue to exist according to the law at the date of making the location. Locations made after the passage of such a law would be subject to the extralateral rights of claims previously located, while the owner thereof would have no compensatory right of pursuing his vein or deposit beyond vertical planes through his boundaries."

While a few minor alterations in the arrangement of certain portions of the work might be suggested, the treatise as a whole reflects great credit upon the author for his painstaking research in the dual fields of science and law.

E. L. K.

Integration by Trigonometric and Imaginary Substitution. By C. O. GUNTHER, M.E., Assistant Professor of Mathematics and Mechanics in the Stevens Institute of Technology, with an Introduction by J. BURKITT WEBB, C.E., Professor of Mathematics and Mechanics in the Stevens Institute of Technology. D. Van Nostrand Company, New York, 1907, pp. 79. Price $1.25 net.

This little book, which has been written in the light of experience, merits the attention of practicing engineers and of those who teach mathematics to students of engineering. It is an attempt to minimize the student's dependence on those terrifying cumbrous things known in the integral calculus as "reduction formulæ" and to qualify the engineer to dispense with the use of tables of integrals. No doubt tables of integrals, like tables of logarithms, will continue to be used as a convenience for saving both energy and time, but this fact does not argue

that those who use them should not be able to construct such tables or to dispense with them as occasion may require.

The authors aim to accomplish their purpose by extending the use of familiar trigonometric principles and trigonometric transformations beyond the limits commonly set for such use in the current text-books. An introduction of 26 pages by Professor Webb gives an interesting general view of the principles involved. This is followed by two chapters in which Professor Gunther deals admirably, by means of examples, with "trigonometric differentials" and "rationalization by trigonometric substitution." Cross cuts are rendered possible by the bold but quite allowable use of the imaginary unit. The book is not designed to replace the usual text-books but is to be used in conjunction with them after the student has acquired the notions of integration and integral and has had some practice in the matter. Exercises are freely introduced and a table of answers is appended.

C. J. KEYSER.

Columbia University Quarterly, Vol. X., No. 1, December, 1907. Published by the Columbia University Press.

This number of the *University Quarterly*, which is doubtless in the hands of many of our readers, is devoted largely to the Schools of Mines, Engineering and Chemistry. It contains a sketch of the history of the School of Mines, by Professor Chandler, and papers on the relations of technical education to industry by Charles B. Going, '82, on the department of civil engineering by Professor Burr, on the testing laboratory by Professor Woolson, on the department of electrical engineering by Professor Crocker, and on the development of mechanical engineering at Columbia by Professor Lucke.

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Journal of Applied Science

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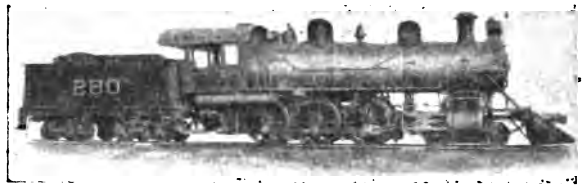

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HIGH SPEED ELEVATORS.*

BY W. W. LIGHTHIPE, E.E.

Vertical transportation in New York City has reached enormous proportions, and the casual observer can hardly realize that there are, approximately, twice as many people carried vertically as are carried horizontally every twenty-four hours. This fact, however, is shown by the following interesting statistics.

We learn from the records of the Building Department that there are at present in the Borough of Manhattan alone approximately 9,000 passenger elevators, of which about 1,000 are located in private residences. In addition to these there are approximately 12,000 freight or service elevators which are used for carrying people or merchandise; thus making a total of 21,000 in the Borough of Manhattan. Four thousand of these elevators were installed in the last six years.

Taking twenty-six of our large office buildings in the lower part of the borough, all of eighteen floors and over, we find a total of five hundred seventy-two floors in all, aggregating a height of approximately one and one third miles, with a rentable area of over five million square feet, or approximately one hundred sixteen acres. In these twenty-six buildings there are one hundred sixteen express elevators travelling an aggregate distance of two hundred seventy-five miles per hour and averaging two hundred forty-three thousand passengers per day of eight hours. These same twenty-six buildings have one hundred fifteen local elevators running ap-

* A paper read before the Electrical Engineering Society of Columbia University, April 10, 1908.

proximately the same number of car miles per hour, but carrying about three hundred seventy-two thousand persons per day of eight hours. This makes a total of two hundred thirty-one elevators running four thousand four hundred miles carrying a total of six hundred fifteen thousand passengers per day.

Taking the eight thousand passenger elevators in the borough and dividing them into groups allowing for the number of people carried in apartment houses, stores, loft buildings, small office buildings of different heights, and large office buildings, we find

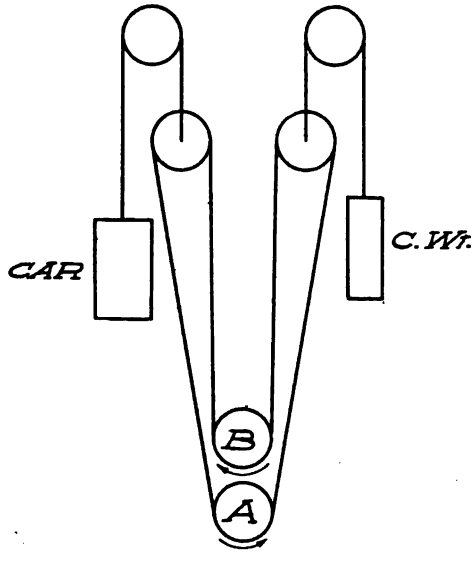


FIG. 1.

that these elevators, in the Borough of Manhattan alone, carry approximately six million five hundred thousand passengers per day. From the last report of the Public Service Commission we learn that only three million five hundred passengers are carried per day by surface, elevated, and subway cars in the entire city of Greater New York.

For a long time the number of high speed hydraulic passenger elevators far exceeded the number of electric elevators of the same class, but elevator manufacturers for some years past have devoted considerable time to developing a high speed electric elevator which would be absolutely safe and more economical to operate. The drum type of electric elevator reached its limit of car speed at

about four hundred feet per minute; speeds above this requiring too high a drum peripheral speed and making it difficult to obtain a high car acceleration and proper retardation. With the large drums the back lash in the gears was objectionable, and although drum-brakes were employed the speed of the car was limited.

The Sprague-Pratt screw type of machine was introduced, but it was only practical to operate the cars at a speed up to four hundred fifty feet per minute, and this type was abandoned.

The duplex motor friction drive machine with endless driving ropes then made its appearance. Fig. 1 shows diagrammatically the arrangements of the driving ropes. *A* and *B* represent the driving sheaves on the armature shafts. The motors are kept in operation continually with armatures turning in opposite directions.

It will be seen that if the driving sheaves are the same size and are turned at the same speed, the endless driving rope will travel at a high speed, but no motion will be transmitted to the car or counterweight. If, however, motor *A* is increased and motor *B* decreased in speed the driving rope on the left will be pulled down by motor *A* faster than it is pulled down by motor *B*, and, therefore, the car will ascend.

If the speed of *A* is decreased and speed of *B* increased the opposite action will result and the car will be lowered. The speeds of the motors are controlled by the fields. This machine allowed a splendid control of the car but was not economical to operate; it was also difficult to get a rope to wear properly. The ropes were spliced and wound with marline to give more friction with the driving sheaves and consequently more tractive effort. Car speeds, however, up to six hundred feet per minute were obtained.

The next machine on the market was a 2:1 traction machine which made use of a slow speed motor, running at approximately one hundred twenty revolutions per minute.

Fig. 2 shows diagrammatically the roping of this machine. One end of the driving ropes was anchored at *C* and the necessary friction with the driving sheave *D* connected directly on the armature shaft was obtained by a weight *W* being on the opposite end of the driving ropes. It will be seen that if the driving sheave *D* is revolved clock-wise the traveling sheaves located in counterweight frame will be drawn down, but with a speed equal to one half the peripheral speed of the sheave *D*. As the counterweight frame is

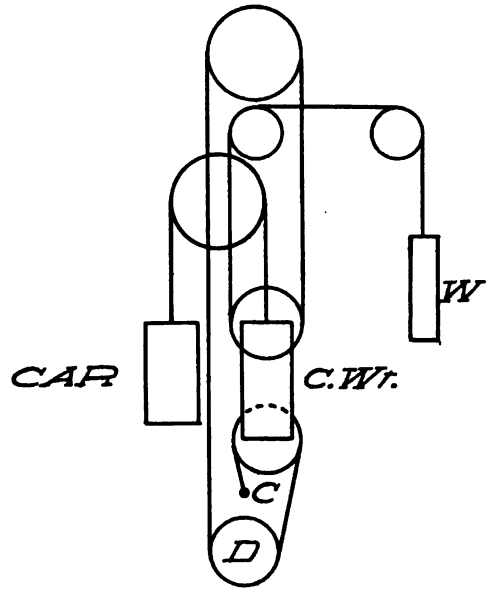


FIG. 2.

connected by rope with the car, the latter will ascend when the former descends. If the driving sheave is turned in the opposite direction the counterweight will, of course, be raised and the car lowered.

The multiplicity of bearings, the excessive space required for overhead sheaves at the top of the building, the uneconomical use of space necessary for machinery, and the excessive lengths of rope were all serious disadvantages. The car, however, could be run at six hundred feet per minute and comparatively high acceleration could be obtained. The kilowatt consumption of the motor was not as low as was expected.

The natural development of this machine resulted in the present 1 : 1 type of traction machine which is now in use in a number of large office buildings.

Fifteen of this type of machine have been recently installed in the Singer Building. Four of these machines operate cars in the tower, one car running the entire distance of approximately 546 feet, from the street level to the fortieth story. Thirty-nine of this same type machine have recently been installed in the new Hudson Terminal Buildings, and are now in operation.

Fig. 3 shows diagrammatically the method of rope drive. Driving sheave *G* is mounted directly on the armature shaft, and the idler sheave *H* is mounted below. Each rope passes from the car over the driving sheave *G* under sheave *H*, again over sheave *G* and to the counterweight. In this way each rope is given two half

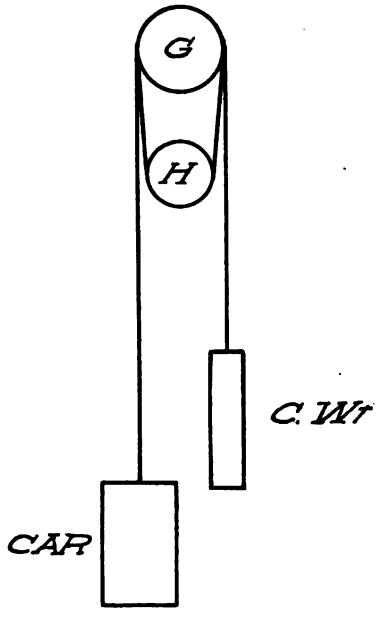


FIG. 3.

turns around the driving sheave, which arrangement results in sufficient traction to drive the car and counterweight. It will readily be seen that when the driving sheave *G* is turned clockwise the car will be raised and the counterweight lowered at a speed equal to the peripheral speed of the driving sheave.

The armature of this machine runs at approximately sixty revolutions per minute to give a car speed of six hundred feet per minute with a thirty-six inch driving sheave, but to do this required approximately a two hundred horse power frame. The motor then develops thirty-five horse power, which enables the carrying of a load of two thousand five hundred pounds at full speed. Less maximum car speed can be obtained by using a smaller driving sheave, but it is not desirable to make the sheave less than thirty inches in diameters when five eighth inch cables

are used. Six cables are generally employed, as shown in Fig. 4, and it will be seen that when the machine is placed at the top of the hatchway the length of each cable will only be equal to the car travel plus the turns around the sheaves and to the counterweight; the latter being at one end of the travel when the car is at the other. The counterweight is usually made equal to the weight of the car plus a certain per cent. of the load to be carried, this per cent. usually being equal to the average load that will be lifted. The load that can be carried by this elevator bears a certain relation to the weight of the car, for the heavier the car and counterweight the greater will be the traction between the driving sheave and the ropes, and the greater the load that can be lifted.

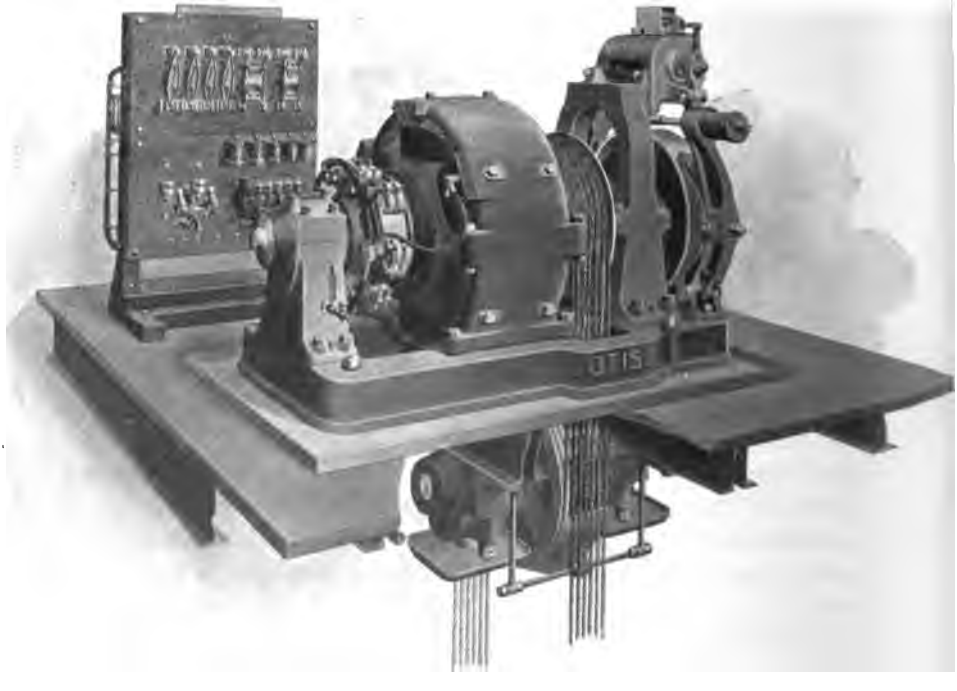


FIG. 4.

On account of the very low armature speed, the armature losses are practically nothing and the absence of all gears makes this a machine of the highest possible efficiency. The greatest advantage of this type lies in its simplicity and safety. A particularly advantageous feature resulting from the arrangement of the cables and the method of driving same is the decrease in traction between

the driving sheave and the cables following the bottoming of either the car or the counterweight, which occurs before the ascending member can strike the overhead work, and the consequent minimizing of the lifting power of the machine until normal conditions are resumed. When the machine is placed overhead all ropes are bent in one direction and are, therefore, less subject to wear. The ropes lead straight to the drive sheave or idler sheave instead of forming an angle with the drum, as is the case with the drum type of machine. There being no reversed bends, the ropes last longer and the maximum strain in ropes is a fixed amount, depending on the traction between motor and ropes.

Ball-bearing shackles are used for making all rope connections, thereby preventing the twisting off of the cables, but allowing them to equalize and to twist and untwist as the car travels from one end of the hoistway to the other.

Each car is provided with a double acting safety device directly connected to a double acting centrifugal governor located at the top of the hoistway, which prevents the car running away in either direction. In the car is an emergency lever which can be thrown by the operator, thus applying the safety on the car by hand at any time. The action of this lever is similar to that of a brake on a trolley car; but in this case the car can be held fast to the guide rails at any desired point in the hatchway. The car and counterweight are each provided with oil buffers at the lower limit of travel. These buffers have been designed to stop the car in a distance equal to a reversed gravity stop, and actual tests have proven them capable of bringing the car safely to rest from full speed. Usually one oil buffer is attached to the counterweight and one is placed in the pit under the car.

Automatic limit switches are placed in the elevator hatchway, in series, near the top and bottom of the hatchway, these switches being opened one after another by the car as it approaches either terminal; by means of these the car is gradually slowed down and brought to rest at the predetermined limits of travel. The switches are independent of any devices placed upon the operating mechanism itself, and are designed to form positive means for bringing the car to rest, whether loaded or empty, at the established limits.

The operator has control of the car with five distinct speeds and can make a full speed reversal of the car or can run at any intermediate speed from fifty to six hundred feet per minute without

any damage to the elevator mechanism. The car is provided with a safety switch, the operation of which by the attendant will open a main line potential switch mounted on the control board, automatically short circuiting the armature through a fixed resistance and applies the brake, thus stopping the car at any desired time, entirely independent of control switch. The throwing of the emergency lever before mentioned also opens the potential switch and stops the machine.

The centrifugal governor located at the top of the hatchway is provided with two switches; the first one would be opened should the car speed increase about ten per cent. above normal. The opening of this circuit increases the field strength of the motor, thus reducing the speed. If, however, the car speed should increase twenty-five per cent. above normal the second switch will be opened, thus opening the potential switch and stopping the machine as described before.

If, for any reason, the car should still increase in speed the governor would operate the safety and the car would be locked to the guide rails.

Quick stops and starts can be made and with the high rate of speed, practically constant at all loads, it is possible to handle more people than with any other kind of elevator with the same maximum speed. The machine has a high efficiency at all loads, practically no momentum of motor armature, on account of the very slow motor speed, and consequently extremely low power consumption, being less than any other type of passenger elevator.

The machine can be placed below, or on any intermediate floor if necessary, in which case the idler sheave is placed directly above the driving sheave. Where it is necessary this machine can be arranged for lifting safes. In that case a magnetic driving sheave is used, thus increasing the cable traction.

A number of the traction type of machine have been in operation for the past four years, some of the cars making as much as thirty miles per day, with a kilowatt consumption per car mile much lower than has ever been obtained before by any type of elevator. Although it is of the simplest possible construction, it is the safest and most efficient, and without doubt the elevator which in the immediate future will be employed in all high class tall buildings.

TOOL STEEL MAKING IN STYRIA.*

By R. F. BÖHLER.

In a metallurgical sense the Austrian Empire is one of the richest countries of Europe, and Styria especially is and always has been the ideal country for tool steel making. It possesses the most extensive and excellent supplies of spathic ores, while its forests furnish charcoal and its mountain torrents abundant water power.

It is only natural that the history of the Styrian steel trade should be a venerable one and its origins be lost in the mists of remotest antiquity.

On the question whether the neolithic age has at all been followed by a bronze period scientists do no more agree; for in Styria we believe and with good reason that there was no such thing. Iron probably was the first metal produced and used by the prehistoric natives of the Styrian Alps.

In Roman times Styrian steel was famous. To quote Swank on this point, "Among the provinces, he says, which contributed largely to the Roman supply of iron was Noricum corresponding to Styria and Carinthia in Austria. Both Pliny and Ovid who lived at the beginning of the Christian era speak of Norican steel as being of superior quality and it is certain that ferrum Noricum was celebrated throughout Italy before their day. The best of swords were made from it in the reign of Augustus, Horace speaks of them. The spathic ores of Styria are still in high favor and the supply of ore especially in the famous iron mountain of Erzberg shows no sign of exhaustion at the end of twenty centuries of almost constant use." The author is a man of exactitude. He says of "almost" constant use, meaning that the ores had a rest from say 500 to 700 A. D. during the great migrations. However, this part of history is very obscure. In the eighth century mining was taken up again at the Erzberg and has not been stopped since.

Throughout the middle ages Styrian steel was considered superior beyond comparison, and was in fact shipped to the

* Read before the New York Academy of Sciences, Section of Astronomy, Physics and Chemistry, November 18, 1907.

remotest corners of the civilized world, which is astonishing, if we stop to think how difficult, slow and unsafe transportation was in those times of the "Holy Roman Empire of German Race."

About 1740 the Sheffield watch maker Huntsman for the first time succeeded in refining steel by melting it in a crucible. He, however, exhibited the "worldly wisdom," as Percy politely styles it, of keeping his process secret. According to the popular tradition of Sheffield knowledge of the process was finally obtained by the "heartless and knavish trick of a rival steel maker" who disguised as a shelter-seeking vagabond, stole the secret and made it the property of the metallurgical world.

Now Styria was the ideal country to take up the new process. For besides its pure ores its charcoal and water power, it moreover possessed both graphite and clay excellently fit for crucible making while other countries get the graphite, for instance, all the way from Ceylon. Raw steel, the material of old fame and produced in the old way on the charcoal hearth was now used for remelting in the newly introduced crucibles and in fact still is today. This accounts for the rather widespread opinion that Styrian steel makers are incurable, sticking to old-fashioned methods and afraid of improvements. This widespread opinion is, it is safe to say, a widespread mistake. They do keep up old methods until the new ones prove superior, but they do not drop them for cheaper ones at the cost of the product. On the contrary Styrian steelmakers have always been the very pioneers of progress. At Urtl, for instance, a blast furnace is recorded to have been in operation since 1567, and puddled steel was regularly produced so far back as 1835. Another instance is the history of tungsten steel. Franz Mayer, of Styria, is assigned the credit of having first applied tungsten to cast steel on a large scale. He exhibited samples of it at the Vienna Congress of 1858 at a time when tungsten-alloyed steel was generally considered a ridiculous experiment. Franz Mayer himself perhaps realized to a moderate degree only the most astounding development tungsten steel was to take. The path he showed us has led to quite unexpected points of view and actually to a new world, metallurgically speaking, the world of high speed work.

Not wishing to dwell upon the tool steelmaking process in general, I shall merely mention a number of special features characteristic of Styria which, it will be seen, are so many special reasons for the superiority of Styrian tool steel.

Take for instance the Kapfenberg works, the largest crucible steel plant in existence, situated about an hour's ride from the old iron city of Leoben of mining school fame. Founded 1446, they are at the same time the most ancient and the most modern. The entire heat treatment throughout the work is under control of an extensive pyrometric system and the latest types of electric hardening, annealing and melting furnaces are in use as well as all of the modern physical and microscopical testing devices. Different from the methods customary in this country the crucibles at Kapfenberg are manufactured on the premises and are used only once, although the majority of them would be able to bear a second or third charging. The reason for limiting their use to one heat only is to be found in the nature of the crucible process, which, far from being a mere remelting operation, involves a multitude of chemical reactions taking place between the iron, carbon, manganese of the raw materials, the carbon and silicon of the crucible material and the different slags formed at the various stages of the melting, as will be seen later.

Most of these reactions vary in degree when the same crucible is used repeatedly, therefore absolute uniformity of the product can only be depended upon when crucibles are used but once. In order to be able to fulfill this great desideratum and furthermore to have absolute control over the chemical composition of the different kinds of crucibles employed for the various qualities of tool- and special steel, the Kapfenberg works include a crucible factory on the premises.

Only very pure graphite can be used for the purpose; when containing impurities to any extent it has to undergo a special purifying treatment. Styrian graphite as used for crucibles contains on an average:

C	SiO_2	Al_2O_3	Fe_2O_3	P_2O_5	K_2O	H_2O
77.80	13.04	6.12	0.44	0.01	0.43	1.95 per cent.

The refractory clay is dried in spacious sheds, then crushed by electrially driven grinding mills, and in the state of a fine dry powder conveyed by means of elevators to the store rooms. Suitable amounts of these materials are then taken through rotating mixers and worked into a paste in a battery of troughs and the plastic mass fed into a set of squeezers. Every squeezer supplies the material for a pair of double working shaping presses from which the crucibles emerge in their final form. The crucible fac-

tory has a capacity of from 12,000 to 15,000 crucibles per week while the drying houses hold 50,000 to 70,000 of them. Careful and gradual drying is indispensable, the crucibles are therefore kept slowly travelling through the drying houses, being shifted at certain intervals from cooler to hotter and drier sections, the entire trip taking six to seven weeks.

The second characteristic feature and perhaps the main cause of the excellence of Styrian steel may be found in the extremely pure ores of the "Erzberg," that is "Ore Mountain." The view (Fig. 1) gives an idea of a small section only, the giant steps or terraces shown go up to the height of 5,000 feet and the whole conical mountain of that peculiar reddish iron ore tint, surrounded by wood-clad and snow-topped mountains reaching 6,000 feet and more, really presents an imposing appearance. Besides being of so great an historical and metallurgical interest, the Erzberg is in fact one of the sights of the Alps. Erzberg ore is almost entirely free from noxious impurities; crude ore yields 38 per cent. to 40 per cent., roasted ore as much as 52 per cent. of iron. Analysis made by the Austrian Government Assay Office average for the crude ore

Fe = 38.9 per cent.
Mn = 2.1 per cent.
P = 0.017 per cent.
S = traces.
Cu = "

At the base of the mountain very silicious ores are found that are smelted in coke blast furnaces, while the higher regions consist of ores low in silica, as given in Table I.

TABLE I. — ANALYSIS OF ERZBERG ORE.

	From Mountain Base.		From Mountain Top.	
FeO, Fe_2O_3	55	53	51	55
MnO	2	3	4	3
SiO_2	16	10	—	—
CO_2	24	26	37	40
MgO, CaO, Etc.	3	8	8	2

While the working of Erzberg is uniform and carried on for common account by the Austrian Alpine Mining Company, the smelting in the charcoal blast furnaces at Vordernberg is under the management of the various owners. The one controlled by the

FIG. 1.

Kapfenberg works is shown in Fig. 2. They all look very much like this one, reminding one of a medieval castle rather than of a sky-scraping coke blast furnace. Also they do not produce quantity but quality.

TABLE II. — COMPARISON OF BLAST-FURNACE PRACTICE.

	Pounds of Charcoal Burned Per Ton of Iron Produced.	
	White Iron.	Gray Iron.
In Styria	1,300	1,500
In Sweden	1,850	2,050
In U. S.	1,640–1,800	2,400

The height of Styrian charcoal blast furnaces varies between 40 and 50 feet, their throat diameter ranges from 3½ to 6½ feet,

FIG. 2.

their bosh diameter from 6½ to 11 feet; the output in 24 hours amounts from 20 to 60 tons. The ratio of fuel to burden varies from 1 : 3 to 1 : 3½ as compared to the ratios of 1 : 2 to 1 : 2½ in general Swedish practice (low figures for gray, high figures for white iron). This is due to the fact that the Styrian siderite is easily reduced and that owing to its peculiar composition only a

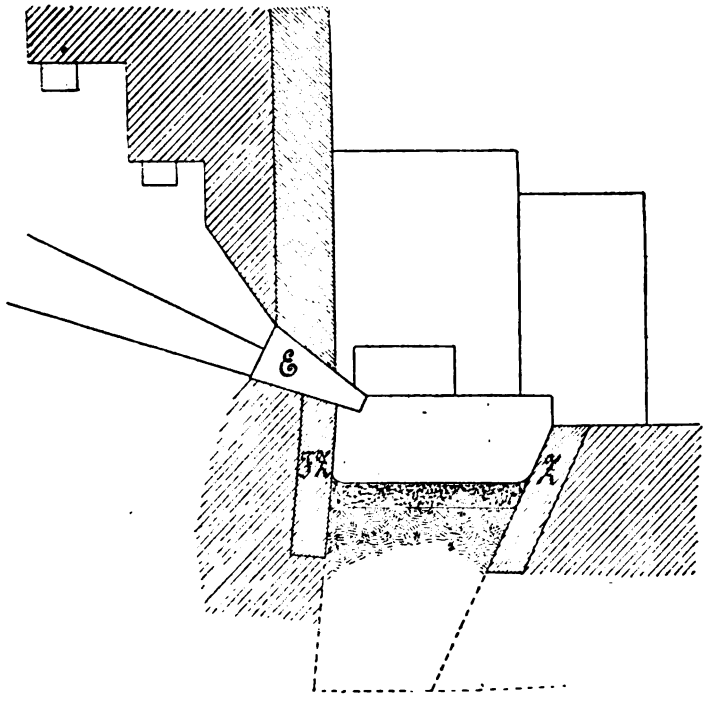
small quantity of flux is needed. For the same reasons the amount of fuel burned per ton of iron produced compares favorably with the figures for American and Swedish practice (see Table II.).

Typical analyses of Styrian charcoal pig iron are shown in Table III.

TABLE III. — ANALYSIS OF STYRIAN CHARCOAL-PIG IRON (LEDEBUR).

	Gray.	Mottled.	White.
C	3.78	3.01	3.72
Graphite	3.07	2.26	——
Mn	2.52	2.33	0.69
Si	1.35	1.79	0.12
S	0.05	0.03	0.02
P	0.03	0.04	0.07
Cu, Co, Ni	0.11	0.05	

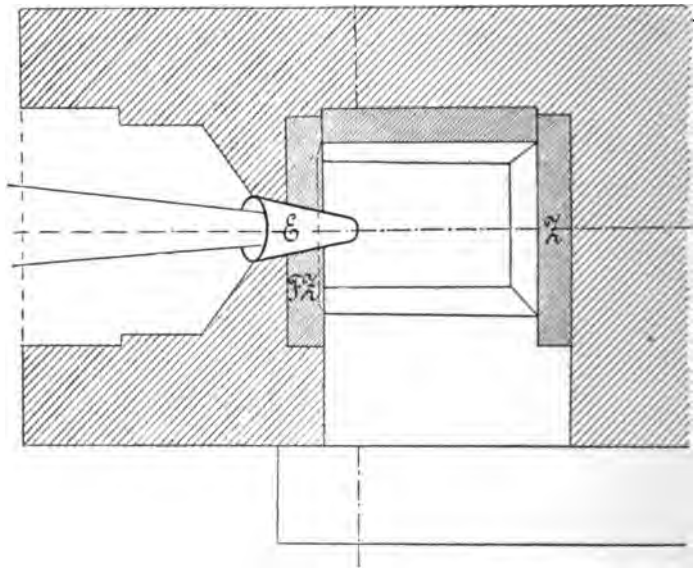
Because of the easy combustion of charcoal the blast is kept at a lower pressure and a lower temperature than is customary in the case of coke. With coke the larger amounts of ashes contained



Refining hearth (section a b).

FIG. 3.

in the fuel and the chemical nature of these ashes make a greater quantity of flux necessary and the slag thus formed has a melting point superior to the temperatures generally reached in charcoal blast furnaces. At the low temperatures of the latter the metal is less liable to absorb S, As, Sb, Cu, Ti, etc., and this fact together with the initial purity of the fuel results in the greater purity of the product.



Refining hearth (Ground plan).

FIG. 4.

The pig iron is converted into steel in the charcoal refinery, and this extensive use of charcoal may be considered another special feature of Styrian steel making. The refining process is a costly one as one may judge by the time it takes; to convert a given quantity of pig iron into steel it takes as many days on the charcoal hearth as it takes hours in the puddling furnace and minutes in the Bessemer converter. Nevertheless this ancient way of refining has been maintained, for the production of a high carbon steel, chemically pure so to speak, is more to be depended upon by this process than by any other.

The oldest apparatus and the one still in common use is the finery fire or hearth (Figs. 3 and 4). It is a flat pit lined on all

four sides with iron plates Z above one of which is placed slanting downwards the tuyere E for conducting the blast. The pit having first been filled with charcoal, the pig iron is placed on it and melted down by the blast.

The chemical reactions underlying the operation are different in character from the other methods of decarburization technically employed. The curves * (Fig. 5) afford a comparison of the general nature of the refining process with the puddling and Bessemer processes. The vertical distances give the amount of the constituents removed in percentages of the amount originally present, and the horizontal distances the time since the commencement of the operations. With gray pig iron the process is carried out in three stages bearing different names. In the first, called "fining," silicon is oxidized into silicic acid, the greatest part of the manganese and part of the carbon. In this way gray iron is converted into white.

The second stage, the "crude refining" involves further oxidation of the carbon partly by the direct action in the air, partly by the indirect action of the oxides contained in the slag. At the end of the second stage the material has acquired the qualities of steel.

Finally the third stage, called "complete refining," consists in the oxidation of the remainder of the carbon, that is to say, in the conversion of steel into wrought iron. Consequently if gray pig iron be used for making wrought iron, the material must be melted down three times, while in the case of white iron the first fining is dispensed with and two meltings suffice. For the production of steel complete refining is not required and two meltings are sufficient in the case of gray iron, or only one in the case of white iron.

On an average the charcoal hearth steel is composed of:

C = 0.800 – 1.300, Mn = 0.043, Si = 0.020, S = 0.005 per cent.
P = 0.015 – 0.019, Cu = 0.004, Co = traces, Ni = traces.

This analysis shows that if the carbon be deducted all the other impurities taken together amount to 0.087 – 0.091 per cent., half of which is manganese. Refined steel is therefore superior in chemical purity to any other kind of raw steel and forms the first and most valuable raw material for the manufacture of Styrian crucible steel.

In the second place puddled steel is used as a further raw

* From Sir Lowthian Bell.

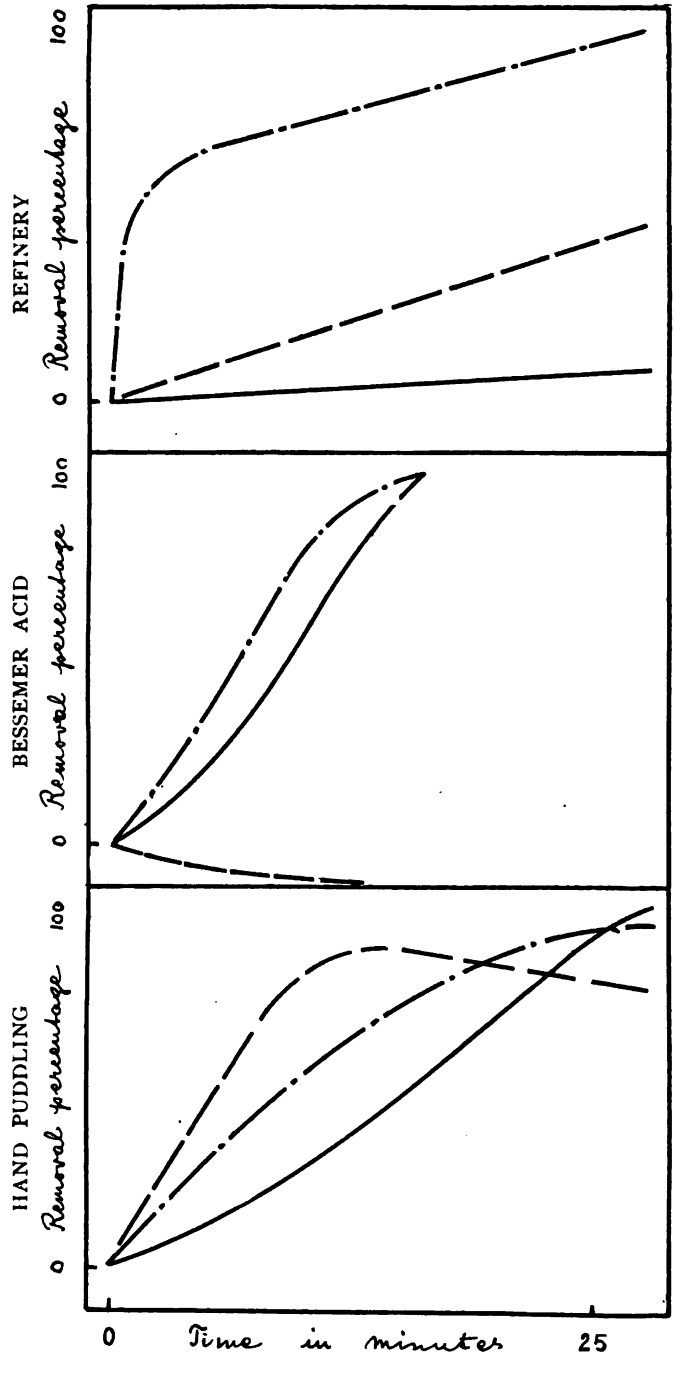

REFINERY
BESSEMER ACID
HAND PUDDLING
Removal percentage
0
100
0
Time in minutes
25
C
Si
P

FIG. 5.

material, mostly in connection with refined steel. It is made from the same charcoal pig iron in puddling furnaces of a special kind with a slight charge. By this method, however, it is not possible to produce steel of such chemical purity as by the finery process, owing to the higher temperatures of the gas furnaces, the greater rapidity of the reactions, and the difficulty of regulating the temperatures at the various stages of the process.

To produce the softest kinds of crucible steel, ingot iron must be used. It is made from charcoal pig iron in a small basic open hearth furnace specially designed for the purpose and yielding a product of extreme purity.

These products then, charcoal pig iron and the materials derived from it in the different ways described above (together with tungsten, chromium, molybdenum, nickel, vanadium, tantalum, uranium, etc., or their alloys in the case of special steels), form the raw materials for the manufacture of Styrian tool steel. In carefully calculated proportions they are charged into the pre-heated crucibles as will be seen presently.

Different from the type in use in this country the modern Styrian crucible furnace is of the overground pattern, allowing an easy handling of the crucibles by means of suspended tongs. One furnace holds forty to fifty crucibles, the object of limiting the number being to insure uniformity of temperature which could not so well be obtained if the furnace were built to hold more. The furnaces are fitted with Siemens regenerators and have a forewarming pit with two separate compartments kept at 400° and 800° C. respectively. The empty crucibles are placed in the first compartment and after remaining there for four hours they are filled with the raw materials and transferred to the second chamber for another period of four hours. They are finally shifted red hot into the melting furnace.

The melting takes from three to four hours; its progress is tested by the foreman with a search iron.

The chemical reactions now taking place in the interior of the crucibles build up one of the most complex problems of scientific metallurgy. Certain amounts of iron oxide are inevitably brought into the crucible on the surface of the pieces of raw steel and pig iron and also by the slag contained in the weld steel. Moreover there is air in the spaces between the material. On melting, the oxides present and those formed under the influence of the enclosed

air first form an oxidizing slag, rich in iron, that acts upon the carbon content of the bath. Carbon monoxide is formed and produces a slight boil. If the crucibles are poor in graphite a decrease in carbon can thus take place during this first period of the process; if, however, the crucibles contain a considerable amount of graphite, no loss of carbon will be noticed in the steel.

During the following period of the melting the slag gradually grows poorer in iron. This is partly due to the carbon in the steel and in the crucible material now acting upon the Fe_3O_4 contained in the slag, partly to the dissolving of crucible material bringing about an increase of the total quantity of slag. The slag by and by loses its oxidizing influence and the graphite laid bare by the dissolving of the crucible wall is absorbed by the steel. At the same time silicon of the clay is reduced and conveyed into the bath.

The temperature is of great influence on these reactions and so is the presence of manganese. The rôle of the latter is particularly complicated. It makes a great difference whether the manganese is contained in the raw materials or added as Mn_3O_4 and at what stage of the process the addition is made.

As to the last stage, the "killing" and the nature of the changes taking place during that time, the views of the different authorities do not coincide. The general belief is that the killing may be due merely to the evolution of gases. It is more likely, however, that killing chiefly acts through enabling the metal to absorb further silicon from the walls of the crucible, thus increasing its solvent power for gas * and thus enabling it to retain in solution during solidification the gas which it contains when molten.

When this stage has been reached the foreman operating the charge will give the signal for the teeming.

Such slight differences of temperature as should happen to exist between the different crucibles are equalized by uniting the forty crucibles of one charge in a ladle previously heated, from the bottom of which the steel is cast into iron molds.

By a special device in the form of a collar of refractory material the pipe due to shrinkage is localized at the very top of the ingot and entirely removed. Every ingot is tested as to physical homogeneity, grain structure and purity of surface, flaws being carefully removed by emery wheels or pneumatic chisels, before the ingot

* Howe, "Metallurgy of Steel," p. 304.

is taken to the hammer plant, or rolling mill. The latter, electrically driven, does quick work and heavy hydraulic presses are in operation for certain purposes, but the greatest part of Styrian tool steel is hammer-forged. Again this process is slow and costly, but the best kneading of the material and thus its finest grain is obtained by it. Such hammer-forging, and the heat treatment and proper manipulation in general appears to be of greater influence on the final quality of high class tool-steel than is often thought to be the case. It requires particular care, a skilled eye and a trained hand and may in fact be called an art. Now in the course of centuries of intimate acquaintance with tool-steel, generations of workmen have arisen in the Styrian Alps with whom this art has become second nature.

In consequence of all of this Styrian steel will be met with everywhere. Besides all kind of tools it is used for scores of other purposes in which highest quality is the main requirement; army rifles, armor piercing shells, shrapnels, shot-proof screens are now made of it and so are the vital parts of recoil field guns, motor cars, steam turbines, etc.

Even in America in the world's greatest of iron-making and iron-using countries, Styrian steel is used to a large extent, being found in the work shops of a number of the largest railroads and imported on a large scale for the United States Army Ordinance Department and other important concerns.*

* For kindly furnished information thanks are due to Messrs. Houghton & Richards, Boston, Mass.

ELECTRODEPOSITION OF NICKEL.*

BY EDWARD F. KERN AND FRANCIS G. FABIAN.

In looking up the literature on the electrodeposition of nickel, it was found that previous to 1890 the principal technical application of the electrolysis of nickel solutions was electroplating metallic articles with a thin coating of nickel. On account of the extreme hardness of the deposit and it not being tarnished under ordinary atmospheric conditions, this industry became quite an important one. Since 1890 most determined attempts have been made to apply electrolysis to the recovery of nickel from solutions produced by leaching ores and mattes, to the extraction of nickel from mattes, speises and alloys, and to the refining of impure metallic nickel.

ABSTRACTS FROM LITERATURE.

I. *Electroplating.*

The electrolytes which have been used for electroplating are: nickel ammonium sulphate, nickel-ammonium chloride, nickel-potassium cyanide and nickel-ammonium phosphate, which in some cases contained varying amount of one or more other chemicals, such as boric acid, phosphoric acid, citric acid, benzoic acid, tannic acid, acetic acid, tartaric acid, or the alkali salts of these acids.† Whichever electrolyte is used, is decided by the fancy of the electroplater, as under proper conditions, such as clean cathode surfaces, neutrality of the solution, purity of solution, current density, temperature of solution, circulation, etc., good, solid, adherent deposits may be obtained from any of these. The usual rule of the electroplater is to vary the conditions until such are found which give satisfactory results, and to adhere to these to the best of his ability. The electrolyte which is in most general use is a slightly acid solution of nickel-ammonium sulphate, the acidity being due to the presence of a sufficient amount of a weak

* Article based on an investigation conducted by Mr. F. G. Fabian in the Metallurgical Laboratory, School of Mines of Columbia University.

† A. Watts, "Electro-Deposition," 1889, pp. 280–330. W. G. McMillian, "Treatise on Electrometallurgy," 1899, pp. 236–249.

acid, such as benzoic, acetic, citric, tannic, tartaric, phosphoric or boric acid, which prevents the formation of insoluble basic salts. The presence of boric acid not only prevents the formation of basic salts, but also causes the deposit to form smoother and less brittle, when the electrolyte contains from 1.5 to 3.0 grams H_3BO per 100 c.c. solution.*

Nickel-ammonium chloride electrolyte, even though a better conductor than one of nickel-ammonium sulphate, is seldom used for electroplating purposes, because of the liability of the deposit to form as spangles if the conditions are not carefully watched.† More recently, however, it was found that very smooth, adherent deposits could be readily produced from neutral nickel chloride electrolytes, but even in this case the deposits were not so satisfactory for electroplating as those obtained from nickel-ammonium sulphate electrolytes, as from the latter, the deposits are harder, consequently are more durable and will wear longer.

A means which has been taken to reduce the E.M.F. required for electroplating from the double sulphate electrolyte is the addition of sodium chloride, 5 to 10 grams per 100 c.c. solution, which, it is claimed, renders the deposit tougher, more reguline, and more adherent, as well as increasing the conductivity of the electrolyte.‡

Two of the difficulties which are often encountered in electroplating are either the formation of gray, pulverulent deposits, or else the deposit does not adhere, but cracks and curls from the cathode. The former difficulty has been found to be due to employing too high current density and too high E.M.F., whereas the latter difficulty is due to the electrolytes being too acid and at too low a temperature,§ or else to a film of grease or dirt on the cathode.

The size and the physical condition of the anode used for electroplating greatly influences the character of the deposit. With a high current density at the anode, gassing results instead of metal going into solution, which causes the electrolyte to increase in acidity. Another cause of gassing is the use of pure metal anodes in nickel-ammonium sulphate electrolyte, as the pure metal does not readily dissolve. However, pure nickel anodes may be successfully used in this electrolyte if a small amount of a chloride

* W. G. McMillian, "Treatise on Electrometallurgy," 1899, pp. 240 and 242.

† *Trans. Am. Electrochem. Soc.*, IX., 1906, p. 220.

‡ A. Watts, "Electro-Deposition," p. 326.

§ W. G. McMillian, "Treatise on Electrometallurgy," pp. 237 and 246.

salt is added, the presence of which causes the anode current efficiency to approximate 100 per cent. * Whether the anode be cast or rolled sheets, the conditions of the electrolysis, as regards the E.M.F. required, will vary accordingly. The cast anodes are more readily soluble than rolled sheets, and also the E.M.F. required for dissolving the former is less than that for the latter. †

II. *Recovery of Nickel from Nickel-Copper Alloys, and Mattes.*

(*a*) The Browne electrolytic process ‡ for treatment of copper-nickel alloys, or mattes, is essentially leaching the granulated material with a dilute hydrochloric acid solution of sodium chloride from the nickel electrolytic cell, the action being accelerated by the presence of chlorine gas. The resulting solution is electrolyzed using an anode of an alloy of copper and nickel, and a cathode of sheet copper. During electrolysis copper is deposited and the solution becomes richer in nickel and lower in copper. Finally when the copper is reduced below 1 per cent., the electrolysis is discontinued and the copper remaining in solution is precipitated by hydrogen sulphide. The hydrogen sulphide is afterwards expelled by boiling, and by current of air. If iron is present, it is oxidized by chlorine, and the solution rendered alkaline with sodium hydroxide. The solution is filtered, evaporated to 50° B., excess of NaCl crystallized out, and the nickel recovered by electrolysis, using sheet nickel cathodes, and carbon rod anodes surrounded by gas-tight porous compartments. The evolved chlorine gas is conducted to the regenerating tower, where it is utilized in digesting granulated matte, or alloy, with the impoverished electrolyte from the nickel electrolytic vats. This process affords a very successful treatment so far as the production of pure metal, but the high cost of electric energy, due to low current efficiency and high E.M.F. in the nickel cells, and to troubles resulting from the use of gas-tight porous anode compartments, the handling of chlorine, and the production of poor deposits, prevents its commercial application.

(*b*) The Ulke electrolytic process § for treatment of copper-nickel

* *Trans. Am. Electrochem. Soc.*, IX., 1906, p. 219.

† W. G. McMillian, "Treatise on Electrometallurgy," p. 240.

‡ "Mineral Industry," X., p. 497. "Electrochemical Industry," I., 1903, pp. 209 and 381.

§ *Trans. Am. Elect ochem. Soc.*, I., 1902, pp. 95–104. "Electrochemical Industry," I., 1903, p. 210. "Mineral Industry," X., p. 498. Report of Canadian Geol. Survey on Nickel and Copper Deposits of the Sudbury Mining District, 1904, p. 223.

alloys is similar to the electrolytic refining of copper, the alloy being made the anode in an electrolyte of mixed copper and nickel sulphates. During electrolysis, nickel accumulates in the electrolyte, replacing an equivalent of copper, which is deposited on the cathode. When the copper is reduced below 1 per cent. the electrolyte is withdrawn from the vat and hydrogen sulphide passes through till all copper is removed. The filtered solution is boiled to expell hydrogen sulphide, and ammonia added till alkaline reaction. The resulting nickel-ammonium sulphate solution is heated to about 60° C. and the nickel recovered by electrolyzing with lead anodes and sheet nickel cathodes. Claims are made that nickel of highest quality was produced cheaply by this process.

(*c*) A more recent process for treating copper-nickel mattes is that of Hybinette.* The material is ground, roasted, and leached with dilute sulphuric acid, which dissolves most of the copper and a small amount of nickel. The residue is retreated by a suitable method of roasting and leaching with dilute sulphuric acid, giving a final residue containing 70 or more per cent. of nickel and a small amount of copper. This residue is mixed with salt and a small amount of either HCl or H_2SO_4, reroasted at low temperature to produce nickel and copper chlorides, which are leached out with dilute HCl. The residue is practically nickel oxide, which is reduced with carbon, melted, cast into anodes, and refined electrolytically in a nickel sulphate solution, which is slightly acidified with a weak acid such as boric, phosphoric, or an organic acid. A specially designed diaphragm cell is used for the electrolysis for recovering pure nickel. The nickel and the copper in the leach solutions are recovered as salts by fractional crystallization; the mother liquors are evaporated to dryness, and roasted along with matte for the next treatment.

There are a number of other processes,† embodying the principles of the three processes given, but no record was found of their being successfully applied.

The essential feature of these processes is the recovery of pure metals from either sulphate or chloride solutions by electrolysis, but

* *Elect. Chem. and Met. Ind.*, IV., 1906, pp. 33–34.

† Report of Canadian Geol. Survey on Nickel and Copper Deposits of the Sudbury Mining District, 1904, p. 222. Borchers and McMillian, "Electric Smelting and Refining," 1904, pp. 275–295. Schnabel, "Handbook of Metallurgy," II., 1907, pp. 731–748. *Eng. and Min. Journal*, 81, 1906, p. 372. *Mineral Industry*, XIII., p. 341, and XV., p. 594. *Elect. Chem. and Met. Ind.*, VI., 1908, pp. 203 and 204.

in most cases very little data is given which will enable the production of satisfactory deposits of nickel from either of these electrolytes.

(*d*) The most important process for the treatment of copper-nickel ores is that used by the Oxford Copper Co. at Constable Hook, N. J., and is known as the Thompson-Oxford Process.* The essential feature of the process is taking advantage of the fact that when copper-nickel matte is melted with sodium sulphide, the Cu_2S forms a copper-sodium matte in which NiS is insoluble, and as the two materials have different specific gravities, the heavier nickel sulphide collects as a "Bottom." The "Tops" and the "Bottoms" are separately smelted several times, with salt cake and coke finally producing "Tops" free from nickel, and "Bottoms" free from copper. Each is treated for its metal content. The final metal product of the "Bottoms" is crude nickel anodes which are electrolytically refined in an electrolyte which is kept secret and guarded closely. It is stated by some that the electrolyte used is a double cyanide of nickel and an alkali, and others claim that it is a nickel-chloride electrolyte slightly acidified with a weak acid, and containing 0.001 per cent. glue.

The Balbach Smelting and Refining Co. at Newark, N. J., refined anodes of crude nickel by the Thum process.† The electrolyte used was a nickel chloride solution containing a secret addition from which the nickel deposited, having an average purity of 99.5 + per cent., absolutely free from carbon and sulphur. The chloride electrolyte prevents the contamination of electrolytic nickel with sulphur, which is not the case when a sulphate electrolyte is used.

III. *Production of Solid Adherent Deposits of Nickel.*

There is much difference in opinion as to the cause of the difficulty of producing solid adherent electrolytic deposits of nickel. Some investigators claim that the cause is due to using an alkaline electrolyte, whereas others say that the presence of free acid is necessary to produce solid deposits.

Pfanhäuser‡ conducted a research on the electrolysis of nickel-ammonium sulphate solution and among his conclusions is the

* *Eng. and Min. Journal*, 64, 1897, pp. 8–9. Report of Canadian Geol. Survey on Nickel and Copper Deposits of the Sudbury Mining District, 1904, pp. 218–221.

† *Elect. Chem. and Met. Ind.*, IV., 1906, p. 26.

‡ *Zeit. für Electrochemie*, 1901, 7 (50), pp. 698–710. *Mineral Industry*, X., 1902, p. 498.

statement that an excess of sulphuric acid is essential to prevent oxidation of nickelous to nickelic sulphate at the anode, and by circulating and heating the electrolyte, the nickel is deposited satisfactorily.

Kugel, instead of acidifying his electrolyte with H_2SO_4, used perchloric or perbromic acid † and produced tough, flexible, adherent sheets of nickel of homogeneous non-crystalline structure, and of any required thickness, when the temperature of the electrolyte was over 30° C.

Adams* found that it was impossible to electroplate homogeneous, adherent deposits of nickel from solutions which contained even a trace of nitric acid, whereas from solutions similar in composition, with this exception, the deposits were satisfactory.

W. McA. Johnson in discussing † the "Physical Character of Electrodeposited Metals" advances the hypothesis that electrolytes, containing the metal as a constituent of a double salt, yield smoother and more homogeneous deposits, which he attributes to the higher E.M.F. required for these electrolytes. As an example, he cites the almost universal employment of nickel-ammonium sulphate for electroplating. It is also stated that more satisfactory results are obtained by electrolyzing at low current density and in cold solutions. Hot solutions accelerate the formation of basic salts, "which adhere to the cathode and cause local distortion of the electrochemical field," rendering the deposit porous and brittle, and in the case of thick deposits, "warty and poor." In further defense of cold solutions ‡ he states that "in hot solutions the reaction velocity of basic salts is accelerated, which means that hot electrolytes become turbid, and a turbid electrolyte always means warty, poor deposits." He, however, claims that elevated temperature is an advantage in that it reduces the E.M.F. required, allows the use of a higher C.D., facilitates circulation, retards curling of the deposit, and produces tougher deposits. Curling of the deposit, he attributes to occluded hydrogen, or a nickel-hydrogen alloy, which establishes unequal tension between the inside and the outside layer of the deposited metal, causing it to crack and peel from the cathode. This theory, he said, is evident for the reason that heavy platings of nickel were produced in a gas-free electro-

* SCHOOL OF MINES QUARTERLY, XXIII., 1902, p. 267.

† *Electrochemical Industry*, I., 1903, pp. 212–214.

‡ *Trans. Am. Electrochem. Soc.*, 1903.

lyte, well circulated in a specially designed and patented vacuum electrolyzing cell.

O. W. Brown in an article on "The Efficiency of the Nickel Plating Tank* states that in order to obtain satisfactory deposits from nickel-ammonium sulphate electrolytes, the chemical composition of the solution must be maintained within narrow limits, such as the purity of the solution, which should be neither appreciably acid nor alkaline. These latter conditions may be controlled by using suitable anodes and so adjusting matters that as much nickel goes into solution at the anode as is deposited on the cathode. It was found that the physical condition of the anode effected its solution: cast anodes dissolve with higher current efficiency and require a lower E.M.F. than do rolled sheet anodes; and also smooth cast anodes require a higher E.M.F. and dissolve with lower current efficiency than similar anodes roughened by acid or cast anodes which contain impurities such as copper, iron or tin. If the current efficiency at the anode is not the same as it is at the cathode, the neutrality of the electrolyte alters accordingly. By properly adjusting the current density at both of the electrodes, the current efficiency at each may be kept the same. The results of the experiments show that the satisfactory operation of a nickel electrolyte depends, to a large extent, upon the physical character and purity of the anode and upon the proper adjustment of current density at both the anode and the cathode.

D. H. Browne in an article† entitled "Nickel Cathodes" states that one of the difficulties, encountered by all who attempt to produce sheets of nickel thicker than electroplates, is the tendency of nickel deposits to crack and curl from the cathodes. The art of producing solid adherent deposits of almost any desired thickness consists in carefully controlling and maintaining initially good conditions to the end, such as concentrated pure neutral electrolyte, circulation of electrolyte, constant current density, and constant temperature (better results obtained between 50° and 70° C.). The advantages, gained by using an electrolyte consisting of the more soluble nickel chloride or nickel sulphate, rather than that of the less soluble double sulphate, is strongly emphasized. The conditions which were found to give the best, most flexible and most adherent sheets of nickel from a chloride electrolyte were: neutral solution

* *Trans. Am. Electrochem. Soc.*, IV., 1903, pp. 83-99.

† *Elect. Chem. and Met. Ind.*, I., 1903, p. 348. *Mineral Industry*, XIII., p. 342.

containing 7 grs. $NiCl_2$ and 18 grams NaCl per 100 c.c.; temperature of 60° to 70° C.; C.D. of 10 to 20 amperes per square foot; and efficient circulation.

The essence of Pfanhäuser's investigation of nickel-ammonium sulphate electrolytes * is that dilute solutions and high C.D. are to be avoided; and that circulating and heating the electrolyte equalizes the difference in concentration, which enables a higher C.D. to be employed than when the electrolyte is neither heated nor circulated.

Snowdon attributes † the non-adherence of electrolytically deposited nickel upon nickel, as due to a thin coating of nickel oxide. This he showed to be so, by previously immersing the nickel cathode and an insoluble anode in a *N*/10 HCl electrolyte and passing a current of 2 amperes per square decimeter (19 amps. sq. ft.) at E.M.F. of 3.8 v. and temperature 18° C. After a few minutes running, the copiously evolved hydrogen reduced the nickel oxide film. The cathode was then washed and placed in a nickel ammonium sulphate electrolyte, containing 8 grams $NiSO_4.(NH_4)_2SO_4.6H_2O$ per 100 c.c.; the deposited adhered tightly.

Summary of the Literature.

The electrolyte generally used for electroplating is a solution of nickel-ammonium sulphate, which may be neutral, or else slightly acid with a weak acid such as boric, phosphoric, benzoic, acetic tannic, tartaric or citric; it may also contain alkali salts of these acids. The acid is added for the purpose of dissolving basic salts of nickel, which are the result of oxidation. An excess of acid causes evolution of hydrogen which produces spongy or non-adherent deposits.

Electrolytes containing nickel-ammonium phosphate, are used to less extent than those containing nickel-ammonium sulphate.

In exceptional cases, electrolytes containing nickel chloride, and nickel-potassium cyanide have been used.

The presence of boric acid in nickel electrolytes not only keeps the solution clear, but also causes the formation of a smoother and less brittle deposit than when not present. The amount used was from 1.5 to 30 grams H_3BO_3 per 100 c.c. solution.

* *Mineral Industry*, X., p. 498.

† *Trans. Am. Electrochem. Soc.*, VII., 1905, p. 301. *Elect. Chem. and Met. Ind.*, III., 1905, p. 185.

Nickel deposits formed in nickel-ammonium sulphate electrolytes are harder, smoother and more durable than those from chloride electrolytes.

The addition of 5 to 10 grams of sodium chloride per 100 c.c. of nickel-ammonium sulphate electrolyte reduces the E.M.F., and also causes the deposit to form more reguline, more adherent and tougher, and the anode to dissolve more uniformly.

Pure nickel anodes are less readily dissolved, than those which contain iron, copper, tin, etc., unless the electrolyte contains a chloride.

Rolled sheet anodes are less readily dissolved and require a higher E.M.F. than cast anodes.

Rough cast anodes require less E.M.F., and are more uniformly dissolved than smooth, polished cast anodes, or sheet nickel anodes.

The solutions which are used for recovering nickel from nickel-copper mattes, or alloys, are either dilute sulphuric acid or dilute hydrochloric acid. The copper in either solution is reduced below 1 per cent. by electrolysis and finally completely removed chemically by cementation on metallic nickel, or else by hydrogen sulphide which is afterwards expelled. The purified solution is rendered alkaline, filtered to remove precipitates and the nickel recovered by electrolysis in suitable cells, using a lead anode for the sulphate solution and a graphite anode for the chloride solution.

The refining of nickel is said to be accomplished in electrolytes of nickel-potassium cyanide, or else of neutral nickel-sodium chloride. The sulphate electrolytes are not so satisfactory as these, as the deposits from sulphate electrolytes contain some sulphur.

There is a difference of opinion as to the cause of solid, adherent deposits of nickel; some electrometallurgists claim that the most satisfactory deposits are from alkaline electrolytes, whereas others say that the electrolyte must be acid. The general opinion, however, is that non-adherent deposits are due to: the electrolyte being too acid, the temperature of electrolyte being two low, or else the cathode is not clean. Pulverulent deposits are due to: too high current density, too high E.M.F., and too acid electrolyte.

The formation of insoluble basic salts is prevented by acidifying the electrolyte with a weak acid, such as boric, perchloric, perbromic, or an organic acid, or else with a slight amount of sulphuric or hydrochloric acid.

Conditions which favor solid adherent deposits of nickel are:

slightly acid electrolyte, to prevent basic salts; pure electrolyte; gas-free electrolyte; temperature between 40° and 65° C.; constant current density; low E.M.F.; efficient circulation; clean cathode; and concentrated electrolyte.

Concentrated electrolytes, especially those consisting of very soluble nickel salts, give the most satisfactory deposits, other conditions being equally favorable.

The tendency of nickel deposits to curl is greatest with high current density; said to be due to formation of nickel-hydride, which is the result of greater liberation of hydrogen with high current density and high E.M.F.

The presence of nitric acid in a nickel electrolyte prevents the formation of solid deposits.

The idea which is apt to prevail after one has reviewed the literature on the electrolytic refining of nickel, is that numerous processes have been suggested, devised, and published, but that only a few have gained any degree of technical success. Even the more recent processes possess such disadvantages as large and expensive plants, complex apparatus, and a number of involved operations, which must necessarily entail losses, all of which are quite serious objections.

Solubility of Nickel Salts.

A general statement may be made that the conductivity of an electrolyte varies directly as the weight of salt in solution, and the more concentrated the electrolyte, the more satisfactory the deposit. As examples, may be cited: lead fluosilicate, silver nitrate, gold chloride, copper fluosilicate, nickel chloride, antimony fluoride, and iron chloride, all of which yield more satisfactory deposits than electrolytes of other salts of the respective metals. Another great advantage gained by using the more soluble salt is lessening the inconvenience due to salts crystallizing and collecting between contacts, and also forming on the electrodes, especially the anode, all of which offers high resistance to the passage of the current. So before deciding on what salts would be used in the investigation, the solubility of the more soluble salts of nickel was looked up, and are given in the following table.

*Table of Solubilities of Nickel Salts.**

Nickel bromide: $NiBr_2$; soluble in water, alcohol and ether.

Nickel chloride: $NiCl_2.6H_2O$; easily soluble in water, alcohol and ether.

100 c.c. saturated water solution at 20° C. contains 39 grs. $NiCl_2$.

100 c.c. saturated water solution at 40° C. contains 42 grs. $NiCl_2$.

Nickel iodide: NiI_2; easily soluble in water.

100 c.c. saturated solution at 20° C. contains 60 grs. NiI_2.

100 c.c. saturated solution at 40° C. contains 64 grs. NiI_2.

Nickel cyanide: $Ni(CN)_2$; insoluble in water.

Nickel-potassium cyanide: $Ni(CN)_2.(KCN)_2$; easily soluble in water, more soluble if KCN is present.

Nickel dithionate: $NiS_2O_6.6H_2O$; soluble in 0.897 part water at 12° C.

Nickel-ammonium dithionate: $2NiS_2O_6.9(NH_4)_2S_2O_6.16.5H_2O$; soluble in water and in an excess of ammonia.

Nickel fluosilicate: $NiSiF_6.6H_2O$; easily soluble in water.

Nickel thiosulphate: $NiS_2O_3.6H_2O$; permanent, soluble in water.

Nickel sulphate: $NiSO_4.7H_2O$; soluble in 3 parts water at 20° C. Soluble in 2 parts water at 50° C.

Nickel-ammonium sulphate: $NiSO_4.(NH_4)_2SO_4.6H_2O$; soluble in 12 parts water at 20° C. Soluble in 5 parts water at 50° C.

Nickel-sodium sulphate: $NiSO_4.Na_2SO_4.4H_2O$; soluble in 3 parts water at 20° C.

Nickel-potassium sulphate: $NiSO_4.K_2SO_4.6H_2O$; soluble in 9 parts water at 20° C.

Nickel phosphates; insoluble in water, soluble in dilute ammonia, and in dilute phosphoric acid.

Experimental Part.

The salts which were selected from the above list, for the preparation of electrolytes to be investigated, were: nickel dithionate, nickel fluosilicate, nickel chloride, and nickel sulphate, because of their higher solubility and their cheapness of preparation.

* Dammer, "Handbuch der Anorganischen Chemie." Roscoe and Schorlemmer, "Treatise on Chemistry." Comey, "Dictionary of Chemical Solubilities." Storer, "Dictionary of Chemical Solubilities." Seidell, "Solubilities of Organic and Inorganic Substances."

The object of the investigation was to determine which of these solutions, with what degree of acidity, at what current density and E.M.F., and at what temperature the most satisfactory deposits of nickel could be produced.

Electrolysis of Nickel Dithionate Electrolyte.

A solution of nickel dithionate was prepared by passing a current of sulphur dioxide into a solution containing a calculated amount of hydrogen peroxide and nickel oxide, according to the reaction:

$$2SO_2 + H_2O_2 + NiO = NiS_2O_6 + H_2O.$$

In order to prevent the formation of sulphuric acid and thiosulphuric acid, the solution was kept at a temperature of $-6°$ C. to $-2°$ C., by placing the containing bottle in a salt-ice water-bath. The sulphur-dioxide was passed into the solution for several hours until complete saturation, and the solution analyzed and found to contain 3 per cent. of nickel as dithionate. The generation of the SO_2 was accomplished by allowing H_2SO_4 to drip into a saturated solution of sodium sulphite in an ordinary gas-generator, which furnishes a steady stream of gas, by the reaction:

$$Na_2SO_3 + H_2SO_4 = Na_2SO_4 + H_2O + SO_2.$$

The electrolysis was conducted by taking about 500 c.c. of the solution and electrolyzing in a beaker with a cast nickel anode (contained 92 per cent. Ni, 5 per cent. Fe, C) and a sheet nickel cathode, size of each 2¼ inches by 2½ inches surface exposure. The current density was 15 amperes per square foot, at an E.M.F. of 2.7 v., when distance between the electrodes was 1 inch, and temperature of the solution was 18° C. The result was decomposition of the salt, and the precipitation of free sulphur on the electrodes, especially the cathode. The gassing at each electrode was considerable. Several trials were made, using a fresh lot of solution, and also on solutions which had been electrolyzed for six hours, but as the results were similar to the first run they showed that dithionate of nickel, even though a very soluble salt, is not suitable for an electrolyte.

Electrolysis of Nickel Chloride, Nickel Sulphate and Nickel Fluosilicate Electrolytes.

For the first set of experiments, it was decided to use solutions which contained 8 grams of nickel per 100 c.c., and free acid to

the extent of one half the molecular proportion of the salt in solution.

The nickel chloride electrolyte contained 32.40 grams $NiCl_2 \cdot 6H_2O$ and 10.43 c.c. conc. HCl (sp. gr. 1.20) or 4.69 grams free HCl, per 100 c.c. solution.

The nickel sulphate electrolyte contained 38.25 grams $NiSO_4 \cdot 7H_2O$ and 3.80 c.c. conc. H_2SO_4 (sp. gr. 1.84) or 6.68 grams free H_2SO_4 per 100 c.c. solution.

The nickel fluosilicate electrolyte contained 42.12 grams $NiSiF_6 \cdot 6H_2O$ and 9.84 grams H_2SiF_6 per 100 c.c. solution. This solution was prepared by digesting a weighed amount of ground pure quartz with a measured amount of 40 per cent. hydrofluoric acid in platinum dishes. The amounts of quartz and acid taken were in the proportions as expressed by the equation: $6HF + SiO_2 = H_2SiF_6 + 2H_2O$. The reaction produced sufficient heat to cause boiling, but was prevented from doing so by adding small amount of distilled water, as required. After the reaction was complete and the solution had cooled, it was diluted to about double volume, and the H_2SiF_6 content determined by titrating with a standard solution of KOH, using a few drops of rosolic acid solution as an indicator, as follows: Ten cubic centimeters of the solution were diluted to 200 c.c. and 10 c.c. of this equivalent to 0.5 c.c. of the original solution, were titrated, after diluting to a volume of about 100 c.c. The end point of the titration is represented by the equation:

$$H_2SiF_6 + 6KOH = 6KF + SiO_2 + 4H_2O.$$

The titration was conducted at a temperature above 70° C. and the standard KOH solution added until the solution changed from pale yellow to cherry-red color.

The titration showed that the solution contained 39.50 grams H_2SiF_6 per 100 c.c.

One liter of the 39.5 per cent. H_2SiF_6 was taken for the preparation of the electrolyte. To this was added 136.2 grams of C.P. nickel oxide (NiO), which dissolved and caused the separation of a small amount of flocculent gelatinous silica. Before the solution was diluted to the required volume (1,338 cubic centimeters) it was analyzed for Ni and H_2SiF_6, and was found to contain them in the proper proportion.

The analysis for the H_2SiF_6 content was done as follows: 10 c.c. of the solution were diluted to 200 c.c. and 10 c.c. of this, equiva-

lent to 0.5 c.c. of the original solution, were taken for analysis. It was poured into a neutral solution of potassium acetate in 50 per cent. alcohol, total volume of about 75 c.c. On standing for about an hour, the precipitated K_2SiF_6 had settled out. The clear supernatant solution was poured through a filter and the precipitate washed twice by decantation and once on the filter paper with 50 per cent. alcohol. After the wash solution had passed through, the precipitate and the filter paper were placed in a beaker and 200 c.c. of water added. It was heated to boiling and the hot solution titrated with standard KOH solution, using as indicator a few drops of a solution of rosolic acid. The end point was change in color from pale yellow to cherry-red. The reaction of the titration is represented by:

$$K_2SiF_6 + 4KOH = 6KF + SiO_2 + 2H_2O.$$

The nickel was determined by taking 40 c.c. of the diluted solution, equivalent to 2 c.c. of the original solution, added bromine, and boiled. About 5 grams of K_2SO_4 were then added to precipitate the SiF_6, and the solution diluted to exactly 200 c.c. The clear supernatant solution was filtered, and the nickel content determined in exactly 100 c.c. of the filtrate (equivalent to 1 c.c. original solution) by precipitating with solution of potassium hydroxide. The precipitate was filtered, washed, dried, ignited strongly and weighed as NiO.

Experiment I.

The first preliminary run was made on the three solutions — the chloride, the sulphate and the fluosilicate, using cast nickel anodes containing 92 per cent. nickel, 5 per cent. iron, carbon and trace copper, and thin sheet nickel cathodes, size of each 2¼ inches by 2½ inches, or 5.6 square inches exposure. The anode and the cathode of each cell were held 1 inch apart by means of a small wooden yoke. The three cells were connected in series and electrolyzed at temperature of 20° C., using a current density of 20 amperes per square foot. The results of the run are recorded in Table I.

Experiment II.

The next run was made in order to find out what effect the current density had upon the deposition, and on the E.M.F. Two sets of three cells each were connected in series, the current density in the three cells of one set was held at 10 amperes per square

TABLE I.

1″ BETWEEN ELECTRODES. C.D. OF 20 AMPS. SQ. FT. TEMPERATUBE 20° C.

Time of Run. Minutes.	E.M.F. per Cell. Cell No. 1. $NiCl_2$.	Cell No. 2. $NiSO_4$.	Cell No. 3. $NiSiF_6$.	Remarks.
Start	0.75 V.	1.00 V.	1.18 V.	
5	0.71 "	1.00 "	1.05 "	
15	0.71 "	1.00 "	1.10 "	No. 1 deposit began to curl. No. 2 cathode bright. No. 3 cathode began to darken. Gassing at all three cathodes, most in No. 2, and least in No. 1. No gassing at the anodes.
25	0.68 "	0.95 "	1.08 "	
45	0.66 "	0.95 "	1.09 "	
75	0.66 "	1.00 "	1.12 "	
105	0.66 "	1.02 "	1.13 "	
120	0.60 "	1.05 "	1.18 "	
150	0.64 "	0.98 "	1.20 "	Deposit No. 1 dark and curled, non-adherent; cathode No. 2, very slight deposit. Deposit No. 3 was very black, and finely divided. Much gassing at all cathodes, none at the anodes.

foot, and that in the three cells of the other set was held at 20 amperes per square foot. Each set consisted of one chloride, one sulphate, and one fluosilicate cell, the solutions containing in each case 8 per cent. nickel and a half molecular equivalent of free acid. The results obtained are given in Table II.

TABLE II.

1″ BETWEEN ELECTRODES. TEMPERATURE 18° C. TO 22° C.

Time of Run. Minutes.	C.D. of 10 Amps. Sq. Ft. Cell No. 1. $NiCl_2$.	Cell No. 2. $NiSO_4$.	Cell No. 3 $NiSiF_6$.	C.D. of 20 Amps. Sq. Ft. Cell No. 4. $NiCl_2$.	Cell No. 5. $NiSO_4$.	Cell No. 6. $NiSiF_6$.	Remarks.
Start	0.64 V.	0.88 V.	0.96 V.	0.73 V.	0.98 V.	1.12 V.	
15	0.63 "	0.90 "	0.98 "	0.70 "	0.95 "	1.13 "	Gassing at all cathodes.
30	0.60 "	0.93 "	0.99 "	0.70 "	0.92 "	1.13 "	Cathode No. 4, dark.
60	0.55 "	0.92 "	0.96 "	0.68 "	0.95 "	1.15 "	Cathode No. 1, dull gray.
90	0.52 "	0.92 "	0.97 "	0.72 "	0.99 "	1.20 "	Some SiO_2 separated in cell No. 6. Deposit dark.
120	0.50 "	0.88 "	0.95 "	0.71 "	1.02 "	1.23 "	
150	0.49 "	0.89 "	0.97 "	0.72 "	1.03 "	1.32 "	
180	0.48 "	0.90 "	1.00 "	0.73 "	1.04 "	1.35 "	Deposits No. 1, No. 4 and No. 6, dark and spongy. Deposit No. 3 less dark. Cells No. 2 and No. 4 no deposit.

There was no gassing at any of the anodes, but there was considerable at the cathodes, most at cathode No. 5 and least at No. 1. In cell No. 6, gelatinous silica separated and collected on the cathode and in bottom of the cell, small amount separated in cell No. 3. The deposits formed on cathodes No. 1 and No. 4, were dark color, and finely divided (spongy), No. 4 being more so. Deposits formed on cathodes No. 3 and No. 6 were better than those on No. 1 and No. 4, they were dark colored but more adherent. The cathodes in cells No. 2 and No. 4 remained bright, no metal was deposited on either.

Experiment III.

The next run was made with three sets of three cells each, connected in series. Each set consisted of a chloride, a sulphate and a fluosilicate cell. One set was electrolyzed at 18° to 20° C., the second set at 38° to 41° C., and the third set at 57° to 62° C. The second and the third sets were heated in water-baths.

All of the electrolytes contained 8 per cent. nickel and a half molecular equivalent of free acid. They were electrolyzed with cast nickel anodes (containing 92 per cent. nickel, 5 per cent. iron, carbon), and thin sheet nickel cathodes, size of each 2½ inches by 2¼ inches (5.6 square inches surface exposed), placed one inch apart.

The electrodes were weighed before and after the run, and the anode and cathode current efficiency calculated. The electrolysis was conducted at a current density of 10 amperes per square foot, and at E.M.F. as recorded in Table III. The electrochemical equivalent of nickel was taken as 1.095 grams per ampere-hour, which factor was used for calculating the current efficiencies. The theoretical amount of nickel deposited in three hours by 0.39 amperes is 1.281 grams.

During the electrolysis, gassing occurred at all of the cathodes, but apparently none at the anodes. There was less gassing in the cells at the higher temperatures than in those operated at ordinary temperature. A small amount of gelatinous silica separated in the fluosilicate electrolytes, collecting in the bottom of the cell and on the electrodes, more so on the cathode which was at 18° to 20° C., than on those which were in the heated electrolytes.

A glance at Table III. will show that the low cathode efficiencies were due to the free acid in the electrolytes, and also that this acid was neutralized during the electrolysis. We also see that with

TABLE III.

Time of Run. Minutes.	Temperature 18° to 20° C.			Temperature 38° to 41° C.			Temperature 57° to 62° C.		
	Cell No. 1. $NiCl_2$.	Cell No. 2. $NiSO_4$.	Cell No. 3. $NiSiF_6$.	Cell No. 4. $NiCl_2$.	Cell No. 5. $NiSO_4$.	Cell No. 6 $NiSiF_6$.	Cell No. 7. $NiCl_2$.	Cell No. 8. $NiSO_4$.	Cell No. 9. $NiSiF_6$.
Start	0.58 V.	0.82 V.	0.88 V.	0.42 V.	0.72 V.	0.88 V.	0.22 V.	0.54 V.	0.62 V.
15	0.57 "	0.92 "	0.89 "	0.43 "	0.74 "	0.85 "	0.22 "	0.46 "	0.64 "
30	0.56 "	0.97 "	0.93 "	0.42 "	0.71 "	0.79 "	0.23 "	0.44 "	0.64 "
50	0.53 "	0.88 "	0.89 "	0.40 "	0.64 "	0.75 "	0.19 "	0.38 "	0.56 "
70	0.48 "	0.86 "	0.92 "	0.36 "	0.60 "	0.74 "	0.20 "	0.32 "	0.56 "
90	0.45 "	0.89 "	1.01 "	0.36 "	0.60 "	0.82 "	0.20 "	0.36 "	0.58 "
110	0.44 "	0.92 "	1.06 "	0.37 "	0.68 "	0.81 "	0.21 "	0.42 "	0.62 "
130	0.44 "	0.89 "	1.01 "	0.32 "	0.66 "	0.79 "	0.20 "	0.42 "	0.66 "
150	0.44 "	0 88 "	0.97 "	0.30 "	0.68 "	0.74 "	0.21 "	0.42 "	0.62 "
160	0.46 "	0.89 "	0.98 "	0.28 "	0.66 "	0.73 "	0.22 "	0.42 "	0.61 "
180	0.45 "	0.89 "	0.99 "	0.27 "	0.64 "	0.73 "	0.20 "	0.42 "	0.59 "
Anode dissolved.	1.16 gr.	1.16 gr.	1.01 gr.	1.05 gr.	0.93 gr.	0.81 gr.	1.24 gr.	1 29 gr.	1.28 gr.
Metal deposited.	0.041 "	0.011 "	0.010 "	0.016 "	0.008 "	0.021 "	0.013 "	0.005 "	0.023 "
Anode efficiency.	90.5 %	90.5 %	78.8 %	82.0 %	72.6 %	63.2 %	96.8 %	100.7 %	99.92 %
Cathode efficiency.	3.20 "	0.86 "	0.78 "	1.25 "	0.62 "	1.64 "	1.01 "	0.39 "	1.80 "
Cathode appearance.	Black, spongy.	Dull white.	Brownish-black, spongy.	Black, less spongy than No. 1.	Gray.	Brownish-black.	Dark brown, more adherent than No. 4.	Bright.	Black, adherent.

rise of temperature of the electrolytes, the E.M.F. is reduced, and also that the anode efficiencies are highest in the electrolytes which were at 57° to 60° C. A study of the cathode efficiencies show a rather peculiar thing in that with the chloride and the sulphate electrolytes the cathode efficiencies decrease with rise of temperature; whereas in the fluosilicate electrolytes, the cathode efficiencies increase with rise of temperature.

Experiment IV.

The results of Experiment III. showed that the presence of a large amount of free acid in the electrolyte caused very low current efficiencies, as well as the formation of unsatisfactory deposits, so it was decided to reduce the free acid to one tenth (or 10 per cent.) the molecular equivalent of the salt in solution, still maintaining the nickel at 8 grams per 100 c.c. The chloride electrolytes contained 32.40 grams $NiCl_2.6H_2O$, and 2.2 c.c. conc. HCl (sp. gr. 1.20) or 1.0 grams free HCl, per 100 c.c. The sulphate electrolytes contained 38.25 grams $NiSO_4.7H_2O$ and 0.70 c.c. conc. H_2SO_4 (sp. gr.

1.84), or 1.34 grams free H_2SO_4 per 100 c.c. The fluosilicate electrolytes contained 42.12 grams $NiSiF_6.6H_2O$ and 1.96 grams H_2SiF_6 per 100 c.c. The chloride and the sulphate solutions were prepared by dissolving a weighed amount of the salts in water, adding the calculated amount of acid and diluting to the required volume. The fluosilicate was prepared by adding a calculated amount of C.P. nickel oxide to a measured volume of the 39.5 per cent. H_2SiF_6 and heating to about 45° C., which gave a clear solution. A small amount of silica separated, it was filtered off, and the clear solution analyzed for nickel and H_2SiF_6. It was then diluted so as to contain 8 grams of nickel and 1.96 grams of free H_2SiF_6 per 100 c.c.

The run was made in the same manner as Experiment III., by electrolyzing a cell of each at 20° C., 40° C. and 60° C., with a current density of 10 amperes per square foot, using cast nickel anodes (containing 92 per cent. Ni and 5 per cent. Fe) and sheet nickel cathodes, size 2½ by 2¼ inches, held one inch apart.

TABLE IV.

Time of Run. Minutes.	Temperature 18° to 21° C.			Temperature 39° to 42° C.			Temperature 58° to 62° C.		
	Cell No. 1. $NiCl_2$.	Cell No. 2. $NiSO_4$.	Cell No. 3. $NiSiF_6$.	Cell No. 4. $NiCl_2$.	Cell No. 5 $NiSO_4$.	Cell No. 6 $NiSiF_6$.	Cell No. 7. $NiCl_2$.	Cell No. 8. $NiSO_4$.	Cell No. 9. $NiSiF_6$.
Start	0.70 V.	0.97 V.	0.67 V.	0.46 V.	0.72 V.	0.70 V.	0.35 V.	0.54 V.	0.52 V.
15	0.72 "	0.92 "	0.71 "	0 53 "	0.73 "	0.72 "	0 30 "	0.58 "	0.52 "
30	0.73 "	0.93 "	0.75 "	0.52 "	0.75 "	0.78 "	0.32 "	0.54 "	0.62 "
45	0.76 "	0.89 "	0.73 "	0.54 "	0.71 "	0.75 "	0.30 "	0.52 "	0.60 "
60	0.76 "	0.89 "	0.79 "	0 53 "	0.78 "	0.82 "	0.34 "	0.52 "	0.65 "
90	0.76 "	0.88 "	0.82 "	0 54 "	0.78 "	0.84 "	0.38 "	0.56 "	0.64 "
120	0.73 "	0.90 "	0.85 "	0.54 "	0.81 "	0.86 "	0.37 "	0.56 "	0.64 "
150	0.73 "	0.91 "	0.88 "	0.54 "	0.81 "	0.88 "	0.36 "	0.56 "	0.63 "
180	0.75 "	0.92 "	0.91 "	0.53 "	0.84 "	0.89 "	0.38 "	0.60 "	0.63 "
220	0.76 "	0.92 "	0.92 "	0.53 "	0.85 "	0.92 "	0.37 "	0.58 "	0.63 "
255	0.73 "	0.91 "	0.94 "	0.49 "	0.78 "	0.92 "	0.37 "	0.54 "	0.64 "
Anode dissolved.	2.01 gr.	2.03 gr.	2.05 gr.	1.96 gr.	1.87 gr.	1.59 gr.	2.03 gr.	1.93 gr.	1.63 gr.
Metal deposited.	1.233 "	0.029 "	0.044 "	1.556 "	0.032 "	0.097 "	1.400 "	0.030 "	0.095 "
Anode efficiency.	102.0 %	103.0 %	104.0 %	99.5 %	94.9 %	80.7 %	103.0 %	98.0 %	82.7 %
Cathode efficiency.	62.6 "	1.5 "	2.2 "	79.0 "	1.6 "	4.9 "	71.0 "	1.5 "	4.8 "
Cathode appearance.	Dull gray adherent deposit.	Bright.	Black spongy deposit.	Dull gray adherent deposit.	Bright.	Dark brown deposit.	Dull gray adherent deposit.	Bright.	Dark brown deposit.

The current was held constant at 0.4 ampere and passed through the nine cells connected in series. The E.M.F. of each cell was taken at intervals and is recorded in Table IV.

The theoretical amount of nickel deposited in four and a half hours by 0 4 ampere is 1 97 grams.

The gassing at the cathodes during this run was much less than during Experiment III., but as before the gassing was greatest in the case of the sulphate electrolytes, and less in the chloride electrolytes, which is also shown by the cathode efficiencies in Table IV.

The cathode efficiencies in the chloride electrolytes show quite an increase in efficiencies, showing that neutral solutions should be employed. The great difference in the cathode efficiencies also indicates that sulphuric acid is a stronger acid than either hydrofluosilicic or hydrochloric acid; that is, it is ionized to a much greater degree than the other two acids, and is a better conductor of the current than the nickel salt.

The variation in the E.M.F. of the fluosilicate cells was due to gelatinous silica which separated as the free acid in the electrolyte was being neutralized. It collected on the electrodes and in the bottom of the cells. There was less silica separated in cell No. 3, which was electrolyzed at 18° to 21° C. than in those electrolyzed at higher temperatures. In cells No. 6 and No. 9, some gassing at the anodes occurred, which accounts for the low anode efficiencies and also for the increase in the E.M.F., as well as for the larger amount of silica, due to the decomposition of free H_2SiF_6 by the current at the higher temperatures.

Experiment V.

The following day, the electrolytes which were used for Experiment IV., were electrolyzed, with a new set of cathodes and cleaned anodes, using a current density of 12 amperes per square foot, in the same manner as before. The data of this run is recorded in Table V.

The theoretical amount of nickel deposited in three hours by 0.43 ampere is 1.413 grams.

The gassing at the cathodes was similar in this run to that of the previous run, with the exception of the chloride cells (Nos. 1, 4 and 7) in which case the gassing was less. The appearance of the cathodes was also similar to the previous runs. There was separation of silica in cells No. 3, No. 6 and No. 9.

These experiments show that the presence of free acid prevents the electrodeposition of nickel from acid solutions, and that when such solutions are electrolyzed, the free acid will be neutralized by the dissolving of the anode and the liberation of an equivalent of hydrogen at the cathode. They also show that as the free acid is being neutralized the E.M.F. becomes greater, especially so in the electrolytes at ordinary temperature, whereas when the electrolytes are heated the increase in the E.M.F. is much less as the solutions become neutral.

TABLE V.

Time of Run. Minutes.	Temperature 19° to 21° C.			Temperature 39° to 41° C.			Temperature 58° to 63° C.		
	Cell No. 1. $NiCl_2$.	Cell No. 2. $NiSO_4$.	Cell No. 3. $NiSiF_6$.	Cell No. 4. $NiCl_2$.	Cell No. 5. $NiSO_4$.	Cell No. 6. $NiSiF_6$.	Cell No. 7. $NiCl_2$.	Cell No 8. $NiSO_4$	Cell No. 9. $NiSiF_6$.
Start	0.81 V.	0.90 V.	0.96 V.	0.50 V.	0.75 V.	0.85 V.	0.42 V.	0.60 V.	0.82 V.
15	0.83 "	0.91 "	0.98 "	0.51 "	0.78 "	0.88 "	0.44 "	0.63 "	0.86 "
30	0.85 "	0.95 "	1.00 "	0.52 "	0.80 "	0.90 "	0.46 "	0.64 "	0.86 "
60	0.87 "	0.97 "	1.00 "	0.51 "	0.77 "	0.85 "	0.43 "	0.58 "	0.81 "
90	0.90 "	1.00 "	1.04 "	0.55 "	0.80 "	0.92 "	0.44 "	0.60 "	0.83 "
120	0.90 "	1.01 "	1.05 "	0.57 "	0.79 "	0.93 "	0.43 "	0.60 "	0.81 "
150	0.82 "	1.02 "	1.08 "	0.54 "	0.80 "	0.94 "	0.42 "	0.60 "	0.79 "
180	0.90 "	1.02 "	1.11 "	0.54 "	0.81 "	0.93 "	0.46 "	0.63 "	0.81 "
Anode dissolved.	1.40 gr.	1.38 gr.	1.42 gr.	1.40 gr.	1.34 gr.	1.24 gr.	1.41 gr.	1.40 gr.	1.23 gr.
Metal deposited.	0.95 "	0.00 "	0.11 "	1.07 "	0.01 "	0.10 "	1.14 "	0.04 "	0.12 "
Anode efficiency.	99.1 %	97.6 %	100.5 %	99.1 %	94.8 %	87.8 %	99.7 %	99.1 %	87.0 %
Cathode efficiency.	67.2 "	0.0 "	7.8 "	75.7 "	0.7 "	7.1 "	80.7 "	2.8 "	8.5 "
Cathode appearance.	Dull gray, adherent deposit.	Bright.	Non-adherent black deposit.	Dull gray adherent deposit.	Bright.	Dark brown deposit.	Dull gray adherent deposit.	Bright.	Dark brown deposit.

The solutions were analyzed after this run in order to determine the change in composition during the electrolysis. The similar solutions were mixed, and the nickel and the acid content determined:

The nickel chloride electrolyte contained 8.4 grams nickel per 100 c.c. solution and no free acid.

The nickel sulphate electrolyte contained 8.25 grams nickel and 0.58 grams free H_2SO_4 per 100 c.c. solution (4 per cent., or $\frac{1}{25}$, the molecular equivalent of the salt).

The nickel fluosilicate electrolyte contained 9.43 grams nickel

and 1.43 grams free H_2SiF_6 per 100 c.c. solution (4 per cent., or $\frac{1}{25}$, the molecular equivalent of the salt).

Experiment VI.

As the presence of free acid in the electrolytes prevented the satisfactory deposition of nickel, and also caused low current efficiencies, the next run was made with neutral solutions.

The salts of nickel chloride, nickel sulphate and nickel fluosilicate were prepared by slowly evaporating the solutions in a water-bath, and collecting the crystals, which were washed and dried. A precaution in preparing the nickel fluosilicate salt is to evaporate the solution at a temperature below 75° C., as at higher temperatures the salt is decomposed; quite rapidly at boiling.

As the composition of the fluosilicate salt was questioned, an air-dried sample was taken and analyzed. Its composition was found to be $NiSiF_6.6H_2O$.

The electrolytes were prepared by dissolving a weighed amount of the air-dried salts in water and diluting to a volume that contained 8 grams nickel per 100 c.c.

With the exception that the electrolytes were neutral, the conditions of this run were the same as those of Experiments IV. and V. There were three sets of cells; one set was electrolyzed at about 20° C., the second set at about 40° C., and the third set at about 60° C. They were connected up in series and electrolyzed with a current density of about 10 amperes per square foot (0.4 ampere). The size of the exposed part of the electrodes was 2½ by 2¼ inches (or 5.6 square inches) and placed 1 inch apart.

In order to measure the exact amount of current which passed, and to get the efficiency more correctly, a standard copper coulombmeter was placed in the circuit. This consisted of two sheet copper electrodes in an electrolyte containing 15 grams copper sulphate crystals, 5 c.c. conc. H_2SO_4, and 5 c.c. ethyl alcohol per 100 c.c. solution. From this electrolyte, one ampere flowing per minute deposits 0.01976 gram copper, or one ampere-hour deposits 1.1856 grams copper. One ampere-hour theoretically deposits 1.0950 grams nickel.

The data obtained during this run are given in Table VI.

Weight of copper deposited was 1.9755 grams, which is equivalent to 1.8245 grams of nickel.

$$1.1856 : 1.095 :: 1.9755 : x. \qquad x = 1.8245.$$

TABLE VI.

Time of Run. Minutes.	Temperature 19° to 22° C.			Temperature 40° to 43° C.			Temperature 57° to 63° C.		
	Cell No. 1. $NiCl_2$.	Cell No 2. $NiSO_4$.	Cell No. 3. $NiSiF_6$.	Cell No 4. $NiCl_2$.	Cell No. 5. $NiSO_4$.	Cell No. 6 $NiSiF_6$.	Cell No. 7. $NiCl_2$.	Cell No. 8 $NiSO_4$.	Cell No. 9. $NiSiF_6$.
Start	0.84 V.	1.42 V.	1.28 V.	0.61 V.	1.33 V.	1.03 V.	0.47 V.	0.82 V.	0.87 V.
15	0.81 "	1.44 "	1.30 "	0.58 "	1.33 "	1.02 "	0.48 "	0 82 "	0.87 "
30	0.81 "	1.51 "	1.31 "	0.58 "	1.35 "	1.01 "	0.45 "	0.79 "	0.84 "
60	0.76 "	1.48 "	1.23 "	0.58 "	1.35 "	1.00 "	0.44 "	0.75 "	0.69 "
90	0.76 "	1.44 "	1.05 "	0.58 "	1.35 "	0.95 "	0.43 "	1.02 "	0.66 "
120	0.76 "	1.44 "	1.06 "	0.58 "	1.34 "	0.94 "	0.43 "	0.87 "	0.68 "
150	0.76 "	1.45 "	1.05 "	0.57 "	1.34 "	0.93 "	0.43 "	0.89 "	0.67 "
180	0.76 "	1.50 "	1.08 "	0.57 "	1.38 "	0.93 "	0.44 "	0.96 "	0.70 "
210	0.76 "	1.50 "	1.06 "	0.60 "	1.38 "	0.92 "	0 44 "	0.96 "	0.70 "
240	0.76 "	1.66 "	1.05 "	0.60 "	1.37 "	0 84 "	0.43 "	0.88 "	0.70 "
255	0.75 "	1.64 "	1.03 "	0.60 "	1.38 "	0.83 "	0.44 "	0.92 "	0.70 "
Metal deposited.	1.761 gr.	1.865 gr.	1.681 gr.	1.814 gr.	1.958 gr.	1.685 gr.	1.809 gr.	1.844 gr.	1.639 gr.
Current efficiency.	96.6%	102.2 %	92.1 %	99.4 %	106.9 %	92.4 %	99.2 %	101.1 %	89.8 %
Cathode appearance.	Dull white adherent deposit, smooth. Cracked from plate.	Black deposit. Peeled from plate. Coated with basic salts.	Bright tough adherent deposit. Solid.	Dull white, finely granular deposit. Adherent.	Black and crystalline non-adherent deposit coated with basic salts.	Very bright, tough and adherent deposit. Solid.	Dull white, finely crystalline, smooth, adherent deposit.	Black crystalline deposit. Peeled from plate and coated with basic salts.	Very bright, tough, non-crystalline deposit. Adherent, solid.

During the electrolysis there was scarcely any gassing at the electrodes in the chloride electrolytes, and extremely little in the other electrolytes, however more in the sulphate electrolytes than in the fluosilicates. In the sulphate electrolytes the gassing seemed to increase as the electrolysis progressed, and also the E.M.F. increased, whereas the E.M.F. of the chloride and the fluosilicate cells decreased.

The apparently very high current efficiencies of the three sulphate electrolytes was due to basic salts which separated during the electrolysis and gathered in crevices, which were due to the splitting of the deposits. The basic salts being insoluble in water, could not be removed by washing. There was also considerable of the basic salts separated from the solution and collected in the bottom of the cell.

The deposits from the chloride and the fluosilicate electrolytes were very satisfactory. The deposits from the chloride electrolytes

were of a dull white to light olive-gray color, finely crystalline and coherent, but showed a slight tendency to peel from the cathode in the cold solutions, whereas in the heated solutions the deposits did not crack nor peel, and also were fairly flexible. The deposits from all the fluosilicate electrolytes were bright, non-crystalline, tough, flexible and adherent, and in no case showed any tendency to crack.

During electrolysis a small amount of gelatinous silica separated in the fluosilicate electrolytes, and settled to the bottom of the cells, more in the one which was heated to 60° C. than in the others. The solutions remained clear.

Some basic salts formed in the chloride electrolytes, but did not interfere with the deposition, as there was such a small amount.

The impurities in the nickel remained as black finely divided sludge which adhered and retained the original shape of the anodes.

Experiment VII.

This run was conducted with a current density of about 20 amperes per square foot, the nine cells and the copper coulombmeter being arranged in series; two sets of the cells were heated in a water-bath similar to the previous runs. The anodes were cleaned of adhering sludge, and a new set of cathodes used. The electrolytes were neutral solutions containing 8 per cent. nickel, the same ones which were used for previous runs.

The electrolysis was conducted for eight hours with a current of 0.8 ampere (20 amperes per square foot of electrode surface).

The results of the runs are given in Table VII.

Weight of copper deposited was 6.8648 grams, which is equivalent to 6.3402 grams of nickel.

$$1.1856 : 1.095 :: 6.8648 : x. \qquad x = 6.3402.$$

In the three sulphate cells there was considerable gassing at the cathodes, and the separation of a yellow basic salt, which collected on the electrodes and floated in the solution. The deposit formed as a black, spongy, non-adherent mass intermixed with basic salts, so these cells (Nos. 2, 5 and 8) were cut out of the circuit after an hour and three quarters.

The only gassing noticed in the chloride and the fluosilicate electrolytes was a very slight amount in cell No. 1, when the current was first started, but after a few minutes this ceased.

As the electrolysis continued, a small amount of basic salts formed

TABLE VII.

Time of Run. Minutes.	Temperature 18° to 22° C.			Temperature 38° to 42° C.			Temperature 56° to 64° C.		
	Cell No. 1. $NiCl_2$.	Cell No. 2. $NiSO_4$.	Cell No. 3. $NiSiF_6$.	Cell No. 4. $NiCl_2$.	Cell No. 5. $NiSO_4$.	Cell No. 6. $NiSiF_6$.	Cell No. 7. $NiCl_2$.	Cell No. 8. $NiSO_4$.	Cell No. 9. $NiSiF_6$.
Start.	0.93 V.	3.50 V.	1.50 V.	0.67 V.	2.25 V.	1.15 V.	0.60 V.	1.40 V.	0.85 V.
30	0.95 "	3.40 "	1.50 "	0.66 "	2.30 "	1.15 "	0.62 "	1.40 "	0.85 "
60	0.90 "	3.40 "	1.30 "	0.68 "	2.20 "	1.10 "	0.60 "	1.35 "	0.90 "
90	0.98 "	3.45 "	1.45 "	0.70 "	2.35 "	1.15 "	0.65 "	1.45 "	0.95 "
105	1.02 "	Removed.	1.48 "	0.70 "	Removed	1.20 "	0.68 "	Removed.	0.98 "
135	1.18 "	.....	1.50 "	0.80 "	.....	1.20 "	0.68 "	.....	1.10 "
180	1.20 "	.....	1.55 "	0.90 "	.....	1.24 "	0.65 "	.....	1.10 "
240	1.20 "	.....	1.53 "	0.85 "	.....	1.30 "	0.65 "	.....	1.10 "
300	1.20 "	.....	1.45 "	0.75 "	.....	1.30 "	0.65 "	.....	1.05 "
360	1.08 "	.....	1 40 "	0.70 "	.....	1.38 "	0.67 "	.....	1.06 "
420	1.06 "	.....	1.30 "	0.68 "	.....	1.45 "	0.63 "	.....	1.08 "
480	1.03 "	.....	1.45 "	0.65 "	.....	1.27 "	0.60 "	.....	1.06 "
Metal deposited.	5.783 gr.		6.136 gr.	6.007 gr.		6.340 gr.	5.601 gr.		5.729 gr.
Current efficiency.	91.2 %		96.8 %	94.7 %		100.0 %	88.3 %		90.3 %
Cathode appearance.	Dull gray, solid, finely crystalline deposit, cracked from edge of plate.	Black, spongy, non-adherent; mixed with basic salts.	Bright, tough, solid, adherent, smooth deposit.	Dull white, solid, finely crystalline deposit, adherent.	Black, spongy, non-adherent; mixed with basic salts.	Bright, tough, flexible, solid, adherent, smooth deposit.	Dull white, solid, finely crystalline deposit, adherent.	Black, spongy, non-adherent; mixed with basic salts.	Bright, tough, flexible, solid, adherent, smooth deposit.

in the three chloride cells (Nos. 1, 4 and 7), but this floated and did not interfere with the deposits. The deposits in these cells were very satisfactory, forming as solid finely crystalline sheets, which were fairly flexible. The deposit in cell No. 1, towards the end of the run, cracked from the edge of the cathode and bent towards the anode, whereas the deposits in cells No. 4 and No. 6 showed no tendency to peel. The deposit from the colder solution (cell No. 1) was darker than those from the heated solutions (cells No. 4 and No. 7). The brightest deposit from the chloride electrolytes formed in the hottest solution (cell No. 7).

The deposits from the three fluosilicate electrolytes were perfect, being bright, very smooth, flexible and adherent, showing no tendency to peel. The electrolyte underwent slight decomposition, causing the precipitation of gelatinous silica, which floated in the solution and settled to the bottom of the cells. The greater amount formed in cell No. 9, which was heated at about 60° C.

The solutions, however, remained very clear, no basic salts formed.

The impurities in the nickel remained as black sludge, retaining the original shape of the anodes.

Experiment VIII.

Another run was made to see if the addition of sodium salts to the chloride and the sulphate solutions would be beneficial. Those salts were added in one half (or 50 per cent.) the molecular proportion of the nickel salts present (as $NiX_2 + NaX$). The chloride solution contained 32.40 grams $NiCl_2.6H_2O$, and 8.0 grams NaCl per 100 c.c., or 8 per cent. nickel and 3.15 per cent. sodium. The sulphate solution contained 38.25 grams $NiSO_4.7H_2O$ and 21.94 grams $Na_2SO_4.10H_2O$ per 100 c.c., or 8 per cent. nickel and 3.14 per cent. sodium. These solutions were electrolyzed at 20° C. and at 60° C., in the same manner as the previous runs, using the same sort of electrodes, and with a current density of 20 amperes per square foot (0.8 ampere).

The results of the run are given in Table VIII.

TABLE VIII.

Time of Run. Minutes.	Temperature 18° to 22° C.		Temperature 58° to 63° C.	
	Cell No. 1. $NiCl_2+NaCl$.	Cell No. 2. $NiSO_4+Na_2SO_4$.	Cell No. 3. $NiCl_2+NaCl$.	Cell No. 4. $NiSO_4+Na_2SO_4$.
Start	1.05 V.	3.00 V.	0.60 V.	0.90 V.
15	1.05 "	3.00 "	0.60 "	0.90 "
60	1.08 "	3.05 "	0.58 "	0.90 "
90	1.00 "	3.25 "	0.56 "	1.06 "
120	1.00 "	3.15 "	0.56 "	1.02 "
180	0.96 "	3.25 "	0.57 "	1.01 "
Metal deposited.	2.656 grams	0.301 gram	2.936 grams	2.937 grams
Current efficiency.	90.2 %	10.2 %	99.7 %	99.8 %
Cathode appearance.	Dull white deposit, which cracked from the plate, but adhered as curls.	Black solid deposit, which cracked from the plate and adhered as curls.	Dull white deposit, similar to that formed in cell No. 1.	Smooth dull white adherent deposit, solid and no cracks.

The weight of copper deposited was 3 1872 grams which is equivalent to 2.9436 grams of nickel.

$$1.1856 : 1.095 :: 3.1872 : x. \qquad x = 2.9436.$$

The addition of sodium salt to the sulphate electrolyte had the

effect of increasing the conductivity, especially when the electrolysis was conducted at 60° C. The deposit from the heated sulphate electrolyte was very satisfactory, it was a smooth, solid, adherent, dull white deposit free from cracks, and the current efficiency of this deposition approximated 100 per cent. The deposit from the sulphate electrolyte at 20° C. was dark, but solid, and was attached to the plate as a number of small curls. The efficiency of the deposition from the sulphate electrolyte at 20° C. was quite low, due to considerable gassing at the electrodes, as the result of high E.M.F., whereas the E.M.F. of the sulphate electrolyte, which was at 60° C., was below 1.1 volts.

Comparing the E.M.F. of the chloride electrolytes with the E.M.F. of the electrolytes which contained no sodium chloride (Experiment VII., cells No. 1 and No. 7), it is seen that the conductivity is not altered by the presence of the sodium salt, but its presence is beneficial in that the current efficiency is increased in the heated electrolyte.

Electrolysis of Nickel Fluosilicate and Nickel Fluoborate Electrolytes.*

Nickel Fluosilicate Electrolyte.

The deposits of nickel from neutral nickel fluosilicate electrolytes were so very satisfactory, when the electrolysis was continued for three to five hours, that a run for about 115 hours was made in order to see what effect time had upon the character of the deposit. The electrolysis was conducted with a current density averaging 15 amperes per square foot at an E.M.F. of 1.10 to 1.18 volts, and temperature of electrolyte from 40° to 55° C.

The electrolyte was a neutral solution of nickel fluosilicate, containing about 7 per cent. of nickel. The electrodes were two cast anodes, containing 92 per cent. nickel, 5 per cent. iron, carbon and trace of copper, size 3 inches by 4 inches, between which was placed a sheet nickel cathode, size 3½ inches by 4 inches. The space between the electrodes was about ¾ inch.

During electrolysis, some gelatinous silica separated from the electrolyte, but did not interfere with the deposition, as it settled to the bottom of the cell. The solution remained clear.

The corrosion of the anodes was very even, the impurities (C, Fe,

* Experiments conducted by Mr. A. P. Frapwell.

and Cu) remained as adherent sludge, which retained the original form of the anodes.

The deposit was bright, perfectly smooth, flexible and free from apparent crystallization. It formed $\frac{3}{16}$ inch thick and weighed 214 grams.

Nickel Fluoborate Electrolyte.

Very satisfactory deposits of nickel were also produced in nickel fluoborate electrolytes, which were prepared by adding a calculated amount of boric acid crystals to a 40 per cent. solution of hydrofluoric acid. The reaction is shown by the equation:

$$H_3BO_3 + 4HF = HBF_4 + 3H_2O.$$

The neutral nickel solution was prepared from this by adding nickel carbonate until the acid was completely neutralized:

$$2HBF_4 + NiCO_3 = Ni(BF_4)_2 + H_2O + CO_2.$$

The clear blue solution was diluted so as to contain 8 per cent. nickel, and electrolyzed at room temperature, in glass beakers. The electrodes were two cast nickel anodes containing 92 per cent. Ni and 5 per cent. Fe, size $2\frac{3}{4}$ inches by $2\frac{1}{4}$ inches, between which was placed a sheet nickel cathode of same size. The space between the electrodes was about $\frac{3}{4}$ inch.

The electrolysis was continued for about 300 hours with a current density averaging 12 amperes per square foot, and an E.M.F. of 1.10 to 1.25 volts.

The deposit formed $\frac{5}{16}$ inch thick and weighed 290 grams. It was a perfectly satisfactory deposit, being bright, smooth, malleable, solid and adherent, differing from the fluosilicate deposits in that it was not quite so smooth, but was more nodular, especially on the edges.

Several preliminary runs were made by electrolyzing at 40° C. and at 60° C., but were unsatisfactory, as the solutions decomposed, resulting in the precipitation of a light green insoluble salt. At temperatures below 30° C., this salt did not form, and the solution remained clear during the electrolysis, and on standing for several months. The solution was kept in glass bottles for over three months, during which time the glass was not corroded.

Transparent light green crystals of nickel fluoborate were obtained by slowly evaporating the solution at temperatures not exceeding 35° C. At higher temperatures the solution was decomposed with the separation of a light green insoluble salt.

Summary of Experimental Part.

As a means for ready comparison, the average data of the runs are given in Table IX.

In all of the experiments, cast nickel anodes, containing 92 per cent. Ni, 5 per cent. Fe, carbon and trace of Cu were used. The cathodes were sheet nickel placed one inch from the anodes.

Nickel dithionate is not suitable for nickel electrolytes, as it is decomposed with the separation of sulphur. The E.M.F. required for an electrolyte containing 3 per cent. nickel, and at room temperature, was 2.7 volts.

TABLE IX.

Composition of the Electrolytes.	Experiment.	Temperature °C.	C.D. Amperes per Sq. Ft.	Time of Run. Minutes.	$NiCl_2$ Electrolyte.		$NiSO_4$ Electrolyte.		$NiSiF_6$ Electrolyte.	
					Average E.M.F.	Cathode Current Efficiency.	Average E.M.F.	Cathode Current Efficiency.	Average E.M.F.	Cathode Current Efficiency.
8 % nickel and half molecular equivalent of free acid.	III.	20	10	180	0.49 V.	3.20 %	0.89 V.	0.86 %	0.96 V.	0.78 %
		40	10	180	0.36 "	1.25 "	0.65 "	0.62 "	0.78 "	1.64 "
		60	10	180	0.21 "	1.01 "	0.42 "	0.39 "	0.61 "	1.80 "
8 % nickel and $\frac{1}{10}$ the molecular equivalent of free acid.	IV.	20	10	255	0.73 "	62.6 "	0.91 "	1.50 "	0.82 "	2.20 "
		40	10	255	0.52 "	79.0 "	0.78 "	1.60 "	0.83 "	4.90 "
		60	10	255	0.35 "	71.0 "	0.51 "	1.50 "	0.61 "	4.80 "
8 % nickel and $\frac{1}{20}$ to $\frac{1}{25}$ the molecular equivalent of free acid.	V.	20	12	180	0.86 "	67.2 "	0.97 "	0.0 "	1.02 "	7.8 "
		40	12	180	0.53 "	75.7 "	0.79 "	0.7 "	0.90 "	7.1 "
		60	12	180	0.44 "	80 7 "	0.61 "	2.8 "	0.82 "	8.5 "
Neutral solutions containing 8 % nickel.	VI.	20	10	255	0.78 "	96.6 "	1.50 "	102.2 "	1.05 "	92.1 "
		40	10	255	0.59 "	99.4 "	1.35 "	106.9 "	0.94 "	92.4 "
		60	10	255	0.44 "	99.2 "	0.88 "	101.1 "	0.73 "	89.8 "
Neutral solutions containing 8 % nickel.	VII.	20	20	480	1.06 "	91.2 "	3.45 "		1.45 "	96.8 "
		40	20	480	0.73 "	94.7 "	2.30 "		1.24 "	100.0 "
		60	20	480	0.64 "	88.3 "	1.40 "		1.01 "	90.3 "
Neutral solutions containing 8 % nickel and a half molecular equivalent of sodium salt.	VIII.	20	20	180	1.02 "	90.2 "	3.12 "	10.2 "		
		60	20	180	0.58 "	99.7 "	0.96 "	99.8 "		

The presence of free acid in nickel chloride, nickel sulphate and nickel fluosilicate electrolytes cau very low cathode efficiency, whereas the anode efficiency was in most cases over 95 per cent. By continued electrolysis the free acid was neutralized and the cathode efficiency increased.

The electrolysis of neutral chloride and fluosilicate solutions with current densities of 10 and 20 amperes per square foot gave very satisfactory deposits, and high current efficiencies. Neutral sul-

phate solutions were unsatisfactory, as precipitates of insoluble basic salts formed and intermixed with the deposited nickel.

Heating the chloride and the fluosilicate electrolytes decreased the E.M.F., increased the current efficiences and improved the deposition. The most satisfactory temperature was about 40° C.

During electrolysis of nickel fluosilicate solutions, a small amount of gelatinous silica separated, and collected on the bottom of the cells. Heating of these solutions above 75° C. also caused the separation of silica, but very slowly.

A small amount of basic salts separated during electrolysis from the neutral chloride electrolytes, but did not interfere with the deposition. The presence of a small amount of free acid prevented their formation.

The presence of sodium salt in the nickel chloride electrolytes was not beneficial, whereas its presence in the neutral sulphate electrolytes reduced the E.M.F. over 60 per cent., and also caused the current efficiency to approximate 100 per cent. when the temperature of the electrolyte was about 60° C.; at room temperature its presence was of no great benefit.

Satisfactory deposits of nickel were obtained from neutral solutions of nickel fluoborate electrolytes, when the electrolysis was conducted at room temperature and using a current density of 10 to 15 ampheres per square foot. At temperatures above 35° C. the solution was decomposed, causing the precipitation of an insoluble basic salt.

THE RECENTLY APPOINTED HEAD OF THE DEPARTMENT OF METALLURGY.

By ROBERT PEELE.

At their May meeting, the Trustees of the University appointed Arthur Lucian Walker, E.M., one of the leading metallurgists of the country, to be Professor of Metallurgy and head of that department, in the School of Mines.

Mr. Walker graduated from the School of Mines in 1883, since which time he has been continuously engaged in field work. He began his professional career as chemist and assayer to the Old Dominion Copper Mining Co., Globe, Arizona, and in 1885 became assistant superintendent of the company, in charge of their mines and smelter. During a temporary suspension of operations, throughout part of 1886–7, he was engaged in New York in drawing up plans for the smelting of magnetic iron ore concentrates, after which he returned to Arizona, as Engineer to the Silver King Mining Company. This position he occupied until the end of 1887. On the resumption of work by the Old Dominion Company, he was appointed superintendent of that property. After holding the post for over five years he was called east, in 1893, to take the managership of the Baltimore Electrolytic Copper Refining Company. In the course of the ensuing five and one half years, he remodelled and doubled the capacity of this large plant, making for himself a sound reputation, both as a metallurgist and in his administrative capacity.

In 1899, he accepted the position of General Manager of the smelting and refining plant at Perth Amboy, N. J., then owned by the Guggenheim Smelting Co., but which afterward became the property of the American Smelting and Refining Co. After spending seven years in charge of these works, he went to the Pacific Coast, in 1906, to superintend the construction of an extensive plant projected by the American Smelting and Refining Co., near San Francisco. This construction work was stopped, however, by the recent financial crisis. In 1907, Mr. Walker served as a director of the smelting company.

At different times in the past ten years he has patented a number

of improvements in apparatus for casting copper and in the arrangements for electrolytic refining, which have been largely adopted by modern copper plants.

On severing his connection with the American Smelting and Refining Company, early in the present year, Mr. Walker was induced to accept the call from the university and has already begun actively to prepare for entering upon his duties in the Department of Metallurgy. His appointment is a cause for congratulation. Combining a high personal character with unusual professional attainments, he is admirably fitted to give to the department the administrative strength and thoroughness of instruction that are demanded in this important division of the School of Mines. For nearly twenty-five years he has been engaged in the management of large works and brings with him an intimate knowledge of the business, as well as of the technical side of the industry.

ABSTRACTS—ANALYTICAL CHEMISTRY.

By E. WALLER.

Desiccation of Gases. Matignon (*Bull. Soc. Chim.* [4], III., 353). Metallic sodium may be reduced to powder, by grinding in a mortar together with NaCl, pulverized after fusion. The mixture appears as a gray powder when thus prepared and can be used in a U-tube following one charged with soda lime to effect complete desiccation of gases (such as NH_3). Being so exceedingly hygroscopic, the pulverization is best effected in a mortar over the top of which is tied a sheet of rubber having a hole only large enough to admit the handle of the pestle which it should grip closely.

Purification of Mercury. Bettel (C. N., XCVII., 158). The author has found that by using a 2 per cent. solution of KCy with addition at intervals of a little Na_2O_2 the ordinary impurities may be completely removed from Hg. While the base metals are being dissolved (which in a case cited required about four days) but little Au was removed. After that the removal of Au was fairly rapid, diminishing of course in speed as the proportion of Au became less.

Ammonia in Waters. Ronchèse (*Bull. Soc. Chim.* [4], III., 362). In a former paper (vide QUARTERLY, XXIX., 301) the author, has described a method for determining NH_3 in salts by the reaction with formaldehyd. He has found that the method is applicable to sanitary examinations of waters even when the amounts to be determined may be only a few hundredths of a milligram. According to the amount of NH_3 presumed to be present, one takes 1 liter, 500 c.c., 250 c.c. or less. Acidify with dilute H_2SO_4, and evaporate down to about 40 c.c. Cool, and pour into a stand-glass, rinsing with 10 or 20 c.c. of water, add a few drops of phenolphthaleine solution. Nearly neutralize with 4 per cent. NaOH, and finish the neutralization with N/100 NaOH to a faint pink. Add 4 c.c. of neutral formaldehyd (of half the ordinary strength), and titrate with N/100 NaOH, to a faint pink. To the number of c.c. used add 0.1 c.c. for every 3 c.c. used (as a correction). This corrected number multiplied by 0.00017 gives the amount of NH_3 present.

Potassium — Volumetric Estimation as Cobalt Nitrite. Drushel (*Am. J. Sci.*, XXIV., 433). The solution, which should contain not over 0.2 gm. K_2O, and should be free from NH_4 salts, is treated with an excess of sodium cobalti-nitrite, and evaporated to pastiness on the water-bath. Cool, and add 50 to 100 c.c. cold water, and stir well to dissolve the excess of the precipitant. Wash a few times by decantation (cold water) and then transfer to a Gooch filter (asbestos felt). Dilute a measured excess of standard permanganate to about ten times its volume and bring to a boil. Introduce the filtering felt and finally the Gooch, and stir well. After 5 or 6 minutes add 5 to 25 c.c. of H_2SO_4 (1:7)

and allow to stand a short time. Then run in an excess of N/10 $H_2C_2O_4$ solution from a burette, heat until colorless, and titrate with the standard permanganate. The total amount of permanganate used, less that required for the $H_2C_2O_4$ introduced is calculated to K_2O. If the permanganate is exactly N/10, 1 c.c. = 0.000856 K_2O. The N/10 $H_2C_2O_4$ is made by dissolving 7.1066 gms. pure recrystallized $(NH_4)_2C_2O_4 2H_2O$ in water, adding 50 c.c. strong H_2SO_4, and diluting to one liter.

Ca, Mg or P_2O_5 do not appear to interfere. Ba or Sr in the solution seems to give high results.

Ferrous Iron in Minerals and Ores. Knight (*C. N.*, XCVII., 122). Tests have been made to verify or disprove the assertion of Manzelius, of the Swedish Geological Survey, that fine grinding causes a partial oxidation of the ferrous iron of a mineral. It was found to be true. The same mineral when coarsely ground showed 2.67 and 2.68 per cent. Fe_2O_3, when finely ground 3.11 and 3.26 per cent. Fe_2O_3.

Separating Iron from Manganese, etc. Moore and Miller (*J. Am. Chem. Soc.*, XXX., 593). Pyridine when added to $FeCl_3$ solution containing free HCl precipitates the $Fe_2(OH)_6$ completely. Mn under these conditions will not readily oxidize so as to be precipitated. Ni and Co are not precipitated. Al, Cr and Zn are only imperfectly precipitated.

Chromium — Oxidizing. Calhane (*J. Am. Chem. Soc.*, XXX., 770). Tests were made upon the comparative efficiency and convenience in the use of Na_2O_2 as compared with Br in alkaline solution, for oxidizing Cr to chromate in the examination of minerals. Some of the first experiments seemed to indicate a failure on the part of Na_2O_2 to oxidize small amounts of Cr, but a closer investigation showed that in such cases a perchromate was formed. Allowing to stand, or warming the solution, brought the Cr to the condition of chromate. In the case of some solutions containing but little Cr, the Br oxidation was only imperfectly effected.

Zinc Sulfide. McCay (*J. Am. Chem. Soc.*, XXX., 376). It is noted that ZnS or $Zn(SH)_2$ after precipitation by sodium or potassium sulphide, is soluble in an excess of the sulfide to an appreciable extent. The addition of concentrated solutions of mineral salts causes a separation of ZnS but it separates slowly, and in a slimy condition.

Separation of Nickel from Zinc — Electrolytic. Fischer (*Chem. Ztg.*, XXXII., 185). Not over 0.15 gm. of the metals should be present (as sulphates). To the solution are added: 5 gms. $(NH_4)_2SO_4$; 1 to 3 gms. Na_2SO_3; and 30 c.c. NH_4OH (sp. gr. 0.91). The whole is diluted to 250 or 300 c.c. The solution is kept at 90° to 92° C. The current should be at first 1 amp., diminishing to 0.1 amp. Under these conditions, the Ni is deposited in 20 minutes. In the filtrate (sic) the Zn is obtained by oxidizing with H_2O_2 expelling NH_3 by heating, adding 2 to 3 gms. tartaric, and excess of alkali, and then electrolyzing with a current of 2.5 amp. Zn is completely separated in 45 minutes.

Foerster and Treadwell (*Zts. Electrochem.*, XIV., 89) have recently confirmed the assertion of Thiel and Windelschmidt (*Zts. Angew. Chem.*, XIII., 563) that Ni electrolytically separated from solutions

containing sulfites always contains S. The process above described or similar operations serve to separate Ni from Zn but the Ni thus obtained must be redissolved, and redeposited from a solution containing no sulfite.

Mercuric Chloride in Gun Cotton. Moix (*J. Chem. Met. and Min. Soc. So. Afr.*, VIII.). $HgCl_2$ in minute amounts may be used to conceal the defects as to heat test of an inferior cotton. Its combination with the organic material renders it difficult of detection. One method consists in placing the moist cotton in contact with a piece of previously ignited Ag foil, immersing the flask containing this combination in boiling water and drawing air through it, the escaping air being drawn through 2 per cent. H_2SO_4. The Ag is then taken out and made to form the anode in electrolyzing the H_2SO_4 used (4 volts for 2 hours required). After washing and drying, the Hg is volatilized off onto a microscope slide for examination and identification (vide QUARTERLY, XXIX., 305).

Another method, apparently more expeditious and satisfactory, consists in extracting the cotton with hot dilute KI solution. The K_2HgI_4 formed is non-volatile on evaporation, so the solution may be concentrated to convenient bulk, and electrolyzed as before, for two or three hours.

Copper—Volumetric. Jamieson, Levy and Wells (*J. Am. Chem. Soc.*, XXX., 760). The process consists in titrating CuCNS with KIO_3 in presence of excess of HCl. The reaction is:

$$4CuCNS + 7KIO_3 + 14HCl = 4CuSO_4 + 7KCl + 7ICl + 4HCN + 5H_2O.$$

In application to ores, 0.5 gm. of the material is treated with acids (HNO_3, followed by HCl if necessary), H_2SO_3 being eventually added, and the solution evaporated to fumes of SO_3. Then cool, add 25 c.c. cold water, and heat to dissolve all soluble sulfates. Filter, nearly neutralize with NH_4OH and add 10 to 15 c.c. strong SO_2 water. Heat to boiling and add 5 to 10 c.c. of a 10 per cent. solution of NH_4CNS. Stir well, settle, filter, and wash. Place filter paper containing the precipitate in a 250 c.c. flask, add about 5 c.c. of chloroform, 20 c.c. H_2O, and 30 c.c. conc. HCl. Then titrate with KIO_3 solution, shaking vigorously between additions. The end reaction is when the chloroform (at first violet from dissolved I) becomes entirely colorless. To make another titration it is only necessary to pour off the major part of the water (containing filter pulp, etc.) when the chloroform, with some of the aqueous solution remaining, may receive another lot of CuCNS with H_2O and HCl as above prescribed.

The KIO_3 (which must be pure and neutral) should be dissolved in the proportion of some multiple of 5.892 gms. per liter. *E. g.*, 11.784 gms. KIO_3 per liter gives a value of 1 c.c. = 0.002 gm. Cu. When of a satisfactory degree of purity, standardization is unnecessary. In case of doubt, the standard can be readily established by dissolving and testing a known weight of pure Cu.

Great rapidity and accuracy are claimed for this method.

Bismuth—Electrolytic. Metzger and Beans (*J. Am. Chem. Soc.*, XXX., 589). To a solution of the nitrate was added a little phenol-

phthalein, then NaOH until it was just alkaline, when the precipitate was redissolved by acetic, and 2 gms. H_3BO_3 added. Then on heating to 70° or 80° C. and electrolyzing with a rotating cathode, at about 2 volts, a clean, firmly adherent deposit of Bi was obtained. No peroxide whatever was formed on the anode.

Lead in Ores, etc. Technical method. Low (*J. Am. Chem. Soc.*, XXX., 587). The Pb is obtained in the form of sulfate by the ordinary method (treating with HNO_3, etc., and finally evaporating with H_2SO_4). The $PbSO_4$ is dissolved out from the SiO_2, etc., by means of hot solution of $NaC_2H_3O_2$ containing a little free acetic. From this solution it is precipitated by $K_2Cr_2O_7$ (on 0.5 gm. of ore use 10 c.c. of a 5 per cent. solution). Boil for a few minutes, filter, wash once only, then open up the filter, and rinse the $PbCrO_4$ with a hot solution of $H_2C_2O_4$ into the flask, add alcohol, and boil until all the CrO_3 has been reduced, and the precipitate converted to $PbCrO_4$. Filter, and wash 10 times with cold water, treat paper and all in a flask with hot dilute H_2SO_4, and titrate with standard permanganate.

The acetate solution for dissolving consists of a cold saturated solution diluted with an equal bulk of water and 40 c.c. of an 80 per cent. acetic added per liter.

The oxalic solution is made up of: cold saturated solution 1 part, water 3 parts.

The permanganate solution as used for Fe titrations is too strong. 1.5185 gms. $K_2Mn_2O_8$ per liter affords a solution such that 1 c.c. = 1 per cent. Pb.

Antimony Sulphide. "Golden sulphide." Jacobsohn (*India Rubber Jour.*, XXXV., 77). The "golden sulphide" used by rubber manufacturers is submitted to the following tests:

Moisture — Drying at 60° C. to a constant weight.

Free S — Extracted from dried material by CS_2.

Total S — Oxidation with HNO_3, etc., finally precipitating as $BaSO_4$.

S combined with Sb — Distilling with conc. HCl, and passing the H_2S into Ag solution from which Ag_2S is separated and eventually weighed.

Antimony — Oxidizing 0.2 gm. with fuming HNO_3 to SbO_2, weighing up and subsequently expelling the Sb by repeated heating with NH_4Cl. The Sb is calculated from the loss which is assumed to be only SbO_2.

Other mineral matter (usually SiO_2 and $CaSO_4$) constitutes the non-volatile portion from this test. In some cases it may be necessary to take a larger quantity, treat with HNO_3 then with HCl and tartaric, etc., and make a more extended examination.

Antimony and Arsenic in Lead-antimony Alloys. Howard (*J. Am. Chem. Soc.*, XXX., 378). Treat 0.5 to 2 gms. of the alloy (Babbitt, type metal, etc.) with 60 to 70 c.c. conc. HCl in an Erlenmeyer flask, adding a few drops (not over 2 or 3) of conc. HNO_3, keep just short of boiling until the metal has dissolved. Then boil vigorously for a few minutes until the color changes from red to colorless. (If Fe or Cu is present the tint will be straw yellow.) Then pass H_2S at once through the hot solution for about 15 minutes. This precipitates only As, but reduces Sb, Sn, Cu, etc. Then pass a current of air for about 30 min-

utes, which oxidizes all but the Sb. Add a little tartaric, and 60 to 70 c.c. of water, and filter off As_2S_3 together with some $PbCl_2$. Cold water will serve to wash out all the Sb, but for the determination of the As in the precipitate, hot water will be necessary to remove the $PbCl_2$. The hot water washings need not be added to the filtrate intended for Sb determination. This latter is neutralized by adding Na_2CO_3 in small portions at a time. Finish by addition of a slight excess of $NaHCO_3$, and titrate with standard I solution, using starch indicator.

The precipitate of As_2S_3 or $As_2S_5 + S_2$ rinsed back into the flask in which the precipitation was made, using about 20 c.c. of water. Add a little NaOH (5 drops of a 20 per cent. solution), boil, and decant through a filter, wash, add about 20 c.c. of H_2O_2 solution, and boil down to about 20 c.c. Cool; add about 0.1 gm. KI and 20 c.c. conc. HCl. After standing 5 minutes titrate with standard $Na_2S_2O_3$ solution.

Tungsten in Ores. E. Keley and Kendall (*Mining J.*, Feb. 22, 1908). The ore is decomposed in the usual manner by digestion with aqua regia in a flask. After diluting and decanting through a filter, the residue is washed repeatedly with a 5 per cent. solution of NaCl until the washings show no acidity when tested with ethylamine solution and phenolphthalein. Allow as little as possible of the WO_3 or insoluble residue to get onto the filter. When washed, the filter is put into the flask — disintegrated — and a sufficient excess of standard ethylamine solution added. The flask is stoppered up and allowed to stand *in the cold* with occasional shaking until solution of the WO_3 has been effected. The solution is then titrated back with standard oxalic (2.8 gms. per liter) and phenolphthalein indicator. The amount of ethylamine which has been neutralized by the WO_3 serves as a basis for the calculation. The ethylamine solution is made by using 6 gms. of a 33.3 per cent. alcoholic solution of the base in one liter 1 c.c. = about 0.005 WO_3.

Lanthanum — Volumetric. Drushel (*Am. J. Sci.*, XXIV., 197). The La is precipitated from a neutral solution by use of a measured amount of N/10 $H_2C_2O_4$ or by $(NH_4)_2C_2O_4$ with a few drops of acetic. After filtering and thorough washing, the precipitate is treated with H_2SO (1:4), heated to boiling, and titrated with standard permanganate.

Separation of the Rare Earths. James (*Ch. News*, XCVII., 205). Brief suggestions are made as to the methods for decomposing the minerals containing the rare earths. The methods practically start with the treatment of the elements after obtaining them as oxalates. The paper contains much that is useful, but is incapable of brief abstraction.

Rare Earth Oxalates. Baxter and Dudley (*J. Am. Chem. Soc.*, XXX., 563). An investigation made on the oxalate precipitates of Nd, La, Sm, and Y, showed that the precipitate tends to occlude some of the alkaline oxalate used as a precipitant, the tendency increasing with the concentration of the alkali oxalate. $Na_2C_2O_4$ is carried down to a less extent than the K or NH_4 oxalates. The occlusion is greater at high than at lower temperatures. In presence of a strong acid more in amount than is equivalent to the alkali oxalate, the occlusion is almost wholly prevented except in the case of Y. For fairly pure precipitates, a cold dilute solution and the presence of a sufficient quantity of a strong acid are the conditions necessary.

Sulfite-sodium, Detection in Presence of Sulfate and Thio-sulfate. Weston and Jeffreys (*C. N.*, XCVII., 85). $PbSO_3$ is insoluble in $Na_2S_2O_3$ solution, whereas $PbSO_4$ and PbS_2O_3 are both soluble therein. Hence, to 5 c.c. of a solution containing 0.05 to 0.1 gm. of the mixed salts add solution of some soluble Pb salt to complete precipitation. Wash by decantation a few times and then add 5 c.c. of $Na_2S_2O_3$ solution (mol. gm. weight per liter). Filter and wash thoroughly. $PbSO_3$ only should remain. Prove its presence by treating with H_2SO_4 and obtaining the reactions of SO_2 — odor and action on permanganate.

Carbon Dioxide in Mineral and Other Waters. Stransky (*Chem. Ztg.*, XXXII., 100). Introduce a little ether into a graduated cylinder of convenient capacity. Then run in the sample of water through a fine tube, delivering it below the level of the ether which should cover it to the depth of 2 or 3 mm. Then titrate under the ether layer with phenolphthalein indicator, delivering the standard N/10 alkali from the burette, also through a fine tube. The ether prevents the escape of the CO_2 during the operation.

Commercial Silicon. Limmer (*Chem. Ztg.*, XXXII., 42). 0.25 to 0.50 gm. of the sample are placed in a porcelain boat in a glass tube and slow current of dry Cl (free from O) is passed through the tube. When all air (and O) has been driven out the boat is heated up to a moderate red heat. $SiCl_4$, $FeCl_3$ and $AlCl_3$ are volatilized and condense on the cooler parts of the tube. SiO_2, together with carborundum, $MgCl_2$ and $CaCl_2$, will remain in the boat. The boat is cooled in the current of Cl gas before withdrawing for weighing. The operation should be repeated until a constant weight is obtained. One half to one hour usually suffice for crystalline and fused Si, but amorphous Si (which usually contains more SiO_2) may require as much as 3½ hours.

METALLURGICAL ABSTRACTS.

By WILLIAM CAMPBELL.

Antimony.

Metallurgy of Antimony. *E. and M. J.*, 85, 201. Abstract of article by Germot, *Revue des Produits Chimiques*, Dec. 15, 1907. Antimony sulphide melted in a converter and air blown through. Sulphurous acid and antimony produced. Operation continuous by adding more ore. Fumes carried off into chambers, air admitted and oxysulphides and oxide of antimony produced from the antimony sulphide carried over.

Valuation of Antimony Ore. *E. and M. J.*, 85, 124. Gives a list of antimony minerals and their antimony content. Refers to F. T. Havard's article in *Mineral Industry*, XV., 43–48. Gives prices paid for ore. American buyers commonly make a basis of $1 per unit of 22.4 lbs. when the metal is worth 8c. per lb. for ore containing about 50 per cent. Sb with a variation of one third of the difference in the market price for the metal per 2,240 lbs. up or down.

Copper.

Cost of Lake Superior and Montana Copper: J. R. Finlay. *E. and M. J.*, 85, 856. Contrasts conditions, current costs, working expenses, etc., concentrating, smelting and refining.

The Copper and Tin Deposits of Katanga: J. R. Farrell. *E. and M. J.*, 85, 747. S.E. Congo Free State. Union Minière Du Haut Katanga. Three distinct belts, iron, copper and tin. Copper the most important. Expect to smelt at $50 per ton of copper. The tin deposits probably very important. Eight and one half tons of tin were run down in primitive furnace, shipped to London and sold.

The Cananea Consolidated Copper Co.: L. D. Ricketts. *E. and M. J.*, 85, 754. The reduction works have been remodeled. Eight new furnaces purchased 4 ft. × 17 ft. 6 in. at tuyeres. Two new converter stands, barrel type. Eight ft. × 11 ft. 6 in. electrically operated. Three additional stands to be installed. Reverberatory furnace 19 × 100 ft. for flue dust.

The Cananea Blast Furnace: C. F. Shelby. *E. and M. J.*, 85, 841. Gives full description of the new blast furnaces of the Cananea Con. Copper Co. with very many detailed drawings.

Coal-dust Firing for Reverberatory Furnaces: C. F. Shelby. *E. and M. J.*, 85, 541. Discusses some of the prevalent ideas of use of coal-dust in reverberatory smelting. At Cananea clinker formed at and beyond the throat of the reverberatory which is used to smelt blast furnace flue dust. Coke deposited on top of charge and formed an insulating layer. This was remedied by changing shape of nozzles. That waste-heat boilers will ever be used with this method is problematical.

The Deposition of Flue Dust: C. F. Shelby. *E. and M. J.*, 85, 204. A study of flue dust sizes at varying distances from furnaces of Cananea Consolidated Copper Co. Plan and table.

The Takilma Smelter, Oregon: G. Crerar. *E. and M. J.*, 85, 365. Located 43 miles south of Grant's Pass, Josephine Co., Oregon. One furnace 42 × 96 in. 94-inch column. Sample mill has one 7 × 9-in. crusher, 1 set rolls and coffee mill. "Base" ore runs: S 17 per cent., Cu 5.6 per cent., Fe 27.3 per cent., CaO 2.5 per cent., Al_2O_3 7.4 per cent., SiO_2 27 per cent. "Siliceous" ore runs: 8.5 per cent. Cu and 40 per cent. SiO_2. Limestone flux. Slag runs: SiO_2 42.7 per cent., FeO 23.5 per cent., CaO 19.5 per cent., Al_2O_3 12.6 per cent., Cu 0.33 per cent.

Matte Smelting at Ingot, California: W. B. Bretherton. *E. and M. J.*, 85, 443. Great Western Gold Co. Ore averaged 11.6 per cent. iron as sulphide and 14.3 per cent. of zinc as sulphide also. Hot blast low column and very hot top proved successful. Present furnace 150×42 in. 12 tuyeres a side. Ore mixture: 2.81 per cent. Cu, CaO 5.4 per cent., Al_2O_3 5.3 per cent, baryta 7.4 per cent., SiO_2 18 per cent. Slag runs: CaO 19.5 per cent., FeO 18.4 per cent., ZnO 12.0 per cent., Al_2O_3 10 per cent., BaO 9.8 per cent., SiO_2 28.6 per cent., Cu 0.2 per cent. Matte averages: 28 per cent. Cu, 16 per cent. Zn. Of the zinc 41 per cent. enters slag, 11 per cent. to matte and 48 per cent. is votalized. 7 to 7.5 per cent. coke used.

The Douglas Copper Smelter at Fundicion, Mex.: P. E. Barbour. *E. and M. J.*, 85, 303. One 44 × 160 in. furnace, 13 ft. 6 in. column. Tilting-furnace type of converters. System of flues and dust chambers described. Several photographs illustrate the plant.

The New Balaklala Smelter: J. L. Mauch. *M. and Mins.*, 28, 411. Smelter at Coram, Shasta Co., Cal., 12 miles N. of Redding. 3 blast furnaces and one reverberatory. Four 18 ft. dia. McDougal roasters. Sulphide sampler and custom sampler. Diagram showing handling of ore. Equipment nearly complete when work stopped owing to lack of funds.

Copper Smelting at Mammoth Plant: A. S. Haskell. *M. and M.* 28, 392. Illustrated account of the U. S. Smelting, Refining and Mining Co.'s plant at Kennet, Cal. 3 blast furnaces 180 × 42 in., two new ones 50 × 180 in., 16 ft. tapping settlers. Converter building contains 2 stands, 8 Allis-Chalmers, improved 96 × 150 in. shells. 2 seven-ft. Carlin wet pans, 2 Ingersoll-Rand pneumatic tamping machines.

Utah's Largest Copper Smelter: R. B. Brinsmade. *M. and M.*, 28, 306. An illustrated description of the Garfield plant of the American Smelting and Refining Co., at Garfield, Utah.

Recent Developments at Cerro de Pasco, Peru: J. C. Pickering. *E. and M. J.*, 85, 760. Rich copper ore was shipped to England or the smelters at Yauli and Casapalca. Reverberatory process used to concentrate poorer ores (under 30 per cent. Cu) into 50 per cent. matte. In 1902 there were 14 such plants. The construction of the Cerro de Pasco smelter designed by Klepetko at Tinahuanco begun in 1904. Has 3 furnaces 56 × 180 in. 18 ft. settlers, etc. Several 19 × 60 ft. reverberatories in course of construction.

Mining and Smelting at Cerro de Pasco, Peru: C. C. Sample. *E. and M. J.*, 85, 206. Ores contain an excess of silica. Limestone used as flux. Lead as galena runs 6 per cent. or less. Copper 8–25 per cent. Antimony, arsenic and bismuth occur in the ores. Gold is not found in consequential amounts. Ore treatment by patio process described. Early copper smelters — 1897 a small reverberatory used for matting. Others soon built. Poor ores smelted. Matte shipped along with rich ores. First blast furnace at Huaracara. One water jacket: 4 × 2 ft.: three tuyeres a side, furnace. One Brown roaster 90 × 14 ft. Two Turrett roasters 24 ft. dia. Six Guyer kilns for lump ore. Briquette mill of 50-ton capacity. Ores run 20 per cent. Cu. 30–35 per cent. S. 75 oz. Ag. The Tinahuanco Smelter of the Cerro de Pasco Mining Co., at La Fundicion. 12 ore bins each 2,000 tons. Four furnaces 168 × 54 in. Two 16-ft. settlers. Furnaces end to end, settlers between them. Slag granulated. Four converter stands. Power equipment described.

The Construction and Operation of the Modern Reverberatory Furnace: B. H. Bennetts. *Western Chem. and Met.*, III., 141. Information about building, putting in a bottom, warming up, etc.

Method of Handling Matte at Selby, California: J. C. Bennett. *E. and M. J.*, 85, 252. Description with drawings and photos of cars with pressed steel trays, 3′ 9″ × 2′ 3″ × 7″ deep, sides sloping 45°, used as matte moulds.

Movable Converter Hoods: A. H. Wethey. *E. and M. J.*, 85, 100. Describes the hood and flue used at the Butte Reduction Works. Detailed drawings.

Apparatus for Converting Mattes and Speisses: H. Haas. *Mining World*, 28, 479. An invention relating to an improved process of converting mattes, etc. Shell in two sections. Turns by rack and pinion in plane at 90° to that of the ordinary converter.

Three-Thousand-Ton Concentrator: R. B. Brinsmade and R. L. Herrick. *M. and Mins.*, 28, 449. Description with illustrations of the plant of the Boston Con. Mining Co. at Garfield, Utah.

The Six-Thousand Ton Concentrator: R. L. Herrick. *M. and M.*, 28, 262. An illustrated description of the Garfield concentrator of the Utah Copper Co. Operations at mine in Bingham Cañon.

Progress in the Leaching of Copper Ores and the Production of Pure Copper: O. Frölich. *Metallurgie*, 5, 206. Describes his method and apparatus whereby all ores can be treated. Cement copper produced.

The Ammonia-Cyanide Process for Treating Copper Ores with Recovery of Precious Metals. Electrochem and Met. Ind., VI., 128. Describes the Mosher-Ludlow process which depends on the principle that ammonia at ordinary temperatures forms soluble stable compounds with oxides, hydroxides or carbonates of copper, zinc, nickel or cobalt. These compounds readily dissolve in water containing ammonia. Precipitation as oxide or hydrate at boiling point of water, ammonia distilled and reabsorbed by water or boiled out solution for use over and over again.

On the Oxygen Contents, Eutectics, Specific Gravity, Gas Absorption of Various Coppers at the End of the Refining Process as well as the Composition and Quality of the Final Product: W. Stahl. *Metallurgie*, 4, 761. The gas absorption decreases with fall of temperature and impurity of the copper. SO_2, CO and H are absorbed by copper; CO_2 and steam are not. Table gives percentage of O and specific gravity for coppers from different stages in the process.

Gold and Silver.

Proof Gold and Silver: J. W. Pack. *M. and S. P.*, 96, 324. Method used at San Francisco Mint. For gold use metal as pure as possible, assay cornets for instance. Take 12 ozs., dissolve in 36 ozs. nitro-hydrochloric acid. Allow to stand for two days. Decant leaving silver chloride behind. To this latter add water and filter through double filter — add to decanted solution. Evaporate one half or more. Dilute till each ounce of gold is contained in 20 liquid ozs. of solution. Precipitate in a lower jar on sheet aluminium very slowly by running in the solution at 90 drops a minute. Spongy gold. Wash. Add HCl to dissolve any aluminium. Wash. Melt under borax. Proof silver. Dissolve in nitric acid. Dilute till one oz. Ag is in 20 oz. solution. Stand 3–4 days in sun. Siphon off. Add HCl. Settle, decant and wash with hot water. Add a little HNO_3 and precipitate silver on aluminium. Wash. Add dil. HCl and heat. Wash, dry and melt.

The Great Gold Mines, II.: T. A. Rickard. *M. and S. P.*, 96, 161. A number of localities are famous for the production of individual mines — the Waihi, the Mt. Morgan, the Homestake and the Goldfield Consolidated. Gives table of principal mines in America and Asia.

Ore Testing at Salt Lake: E. Gayford. *M. and S. P.*, 96, 134. Describes plant. Plan. Tests in car-load quantities by amalgamation (plates and Pierce amalgamator); water concentration (jigs and tables); magnetic separation (Ding's or Wetherill's); cyanidation (sand and slime treatment, filter-pressing).

The Pierce Amalgamator. E. and M. J., 85, 114. Adapted to treat only fine material. Screens remove the coarse. Consists of a series of vertical riffles of amalgamated copper which extend below the top of a series of lower riffles which are bent back at the top in the direction of the flow. These latter form a small trap for the quicksilver. The water passing down between the riffles keeps the black sand and flour gold agitated, and by sweeping them against the surface of the amalgamated copper riffles the fine gold has a chance to be caught by the plated surface.

The Cyanine Process — A Review: F. W. Traphagen. *Western Chem. and Met.*, III., 181. Main steps in cyanide treatment from Hagen down. Slime filters and patents.

Notes on Preliminary Cyanide Work: H. F. A. Riebling. *Western Chem. and Met.*, III., 172. Several hundred pounds of ore crushed to 20-mesh. Quarter down to 20 lbs. Take assay sample. Determine

amount of lime to neutralize acidity, three-day bottle test. Test for fineness of pulp in glass jar. May be followed by barrel tests or 5- to 10-ton tank runs.

A Cheap Form of Cyanide Plant: C. Hunter. *Bull. 34, I. M. and Met.*, July 18, 1907. Describes a light and portable form of plant used in southern Rhodesia. Cost a little over £300.

The Hardinge Conical Tube Mill: H. W. Hardinge. *Western Chem. and Met.*, IV., 115. Description and illustration of apparatus, method and reasons.

Hendrix Methods of Cyaniding: L. D. Bishop. *Western Chem. and Met.*, III., 187. Describes work of Hendrix agitator, a cylindrical tank with a conical bottom. Hollow shaft in center with propellers to produce upward current. The Montana-Tonopah M. Co. operates six. Hendrix decanting filter described.

Recent Improvements in Art of Slimes Treatment: D. J. Kelly. *Western Chem. and Met.*, III., 153. Describes the Kelly pressure filter.

The Wade Vacuum Filter: E. M. Wade. *Western Chem. and Met.*, III., 163. U. S. letters patent No. 854,972. Continuous process. Revolving drum composed of several chambers, suction at intervals. Slimes fed to outer surface at top.

The Sweetland Filter Press: E. J. Sweetland. *E. and M. J.*, 85, 359. Describes the construction of the press which consists of a series of cast iron frames, supported on a pair of side rods. Filter mats are carried in the upper portion of the alternate frames. Sludge admitted under pressure. Cake formed. Washed and discharged through a 4-inch pipe line.

Cleaning of Filter Press Plates. Austr. Min. J., 1907, 568. Abst. *J. Chem. Met. and Min. S. S. A.*, VIII., 61. At the Kalgoorlie mines the filter press plates become cemented due to the presence of hydrated calcium sulphate in the cyanide solution. This cement was difficult to remove. By heating the plates in an oven, causing a dehydration of the hydrated calcium silicate, the deposit was rendered easy of removal.

A Method of Settling Slimes as Applied to their Separation from Solution in Cyanide Treatment: H. G. Nichols. *Bull. 41, I. M. and Met.*, Feb. 13, 1908. The apparatus consisted of a pyramidal-shaped tank 4 ft. × 5 ft. connecting at the bottom through an 8-in. sq. aperture with a closed box in which a 10-in. belt was made to travel slowly. The belt removed the solid matter over the edge of the closed box.

Treatment of a Concentrate-Slime: A. E. Drucker. *M. and S. P.*, 96, 458. Describes the Drucker Process as applied to heavy mineral or concentrate slime. Experimental plant at Chittabalbie, Korea. The concentrate from the Frue vanners goes to tube-mills with cyanide solution — spitzkasten. Overflow to circular agitator (mullers) with filter base. Air pumped through filter. When complete (20 hrs.), allowed to settle and decanted. Slime remaining contains 30 per cent. moisture. Vacuum-filtered down to 18 per cent. Next wash with 0.25 per cent. solution, then with 0.1 per cent. Drawing of apparatus given.

Caustic Lime: Notes on the Estimation of: E. H. Crogan. *J. Chem. Met. and Min. S. S. A.*, VIII., 57. Two methods in vogue locally, gravimetric and the volumetric sugar process (G. W. Williams). The gravimetric gave higher results. The results show that the sugar method is correct and that the gravimetric gives too high results.

Action of Alkaline Solutions in Cyaniding: B. L. Gardiner. *Monthly Jl. Chamber of Mines of W. A.*, Oct. 31, 1907, p. 915. Abst. *J. Chem. Met. and M. Soc. S. A.*, VIII., 282. Details of experiments on the accumulated sands at the Burbanks Birthday G. M. Addition of NaOH reduced cyanide consumption from 2.2 lbs. to 0.4 lb. and also reduced the extraction from 78.6 per cent. to 21.4 per cent.

The Cyanide Process; Assisting the Solution of Gold in: by Compressed Air: A. F. Crosse. *J. C. Met. and Min. S. S. A.*, VIII., 36. Used a conical vat, with vertical pipe in center through which the pulp in the vat is lifted by compressed air supplied by the "trompe," 2 ft. above the surface of the liquid. It falls over into an outer tube and is discharged through 4 pipes whose orifices are set tangentially so that the pulp receives a circular motion. In this way aeration and circulation are obtained automatically without any machinery.

Use of Compressed Air in Cyanidation: A. Grothe. *Mining World*, 28, 58. Describes the Pachuca Agitating Tank.

Precipitation by Zinc Dust: R. L. Herrick. *M. and Mins.*, 28, 432. Describes apparatus for automatic feed of zinc dust to gold solution at Homestake, Lead, S. D.

Cyaniding Cripple Creek Ores: F. L. Barker. *M. and Mins.*, 28, 423, 481. Cost of treatment in mills, Golden Cycle and U. S. R. and R. Co.'s rates. Freight. Analyses of ores. Isabella, Wild Horse, Blue Flag Mills. Tube mill practice; Standard Plant, Colorado City; Portland Mill.

Cyanide Precipitation and Clean-up at the Portland, Colorado, Mill: J. M. Tippett. *Western Chem. and Met.*, IV., 17. Refers to article in *Mining Reporter*, Oct. 24–31, 1907, by R. Chauvenet on the Portland Co.'s cyanide mill.

Precipitation room 18 rows boxes, 6 compartments each 4 ft. × 4 ft. × 4 ft., pyramidal bottom. 14-mesh wire screw false-bottom. 200 lbs. zinc shavings each. Dipped in 2 per cent. lead acetate solution. Strong solution 0.6 lb. per ton (protective alkalinity 0.5–1.0 lb.) flows through 6 boxes. Weak solution 0.3–0.4 lb. through other six. 500–600 tons of former and 800–1,000 tons of latter flow through in 24 hrs. White precipitate in head compartments — calcium ferrocyanide, zinc hydrate, calcium sulphate, carbonate and fine slime from sand filters. Clean-up described.

Cyanide Practice on Boulder County Ores: F. Leonard. *Western Chem. and Met.*, III., 11. Describes practice at Cash Mill, Magnolia, Colo. Mill ore averages $15. Dodge crusher 7 × 11 to 1 inch. Cornish roughing rolls 12 × 20 in. Impact screens 6-mesh. Bins. Rotary dryer fixed up as roaster. Crude oil fuel. Gold now free and bright. (Amalgamation system being installed.) Vats charged by hand bar-

rows. Charcoal in ore floated off by weak solution (0.03 per cent.) as preliminary wash. Regular stock solution 1.0 per cent., run on covering charge, further eliminates soluble iron salts. A 30-ton vat required 8 tons weak solution, then 7–8 tons standard solution to cover charge, and as much more run on before solution is strong enough to enter zinc boxes. Cyanide consumption two lbs. per ton of ore. Zinc shavings precipitation.

The Montgomery-Shoshone Mill: P. E. Van Saun. *M. and M.*, 28, 385. The new plant of the Bullfrog Reduction and Water Co. at Rhyolite, Nev. 200-ton receiving bin. Ore $30 ton. 24″ × 15″ Blake crusher. 48″ Vezin sampler. 42″ × 14″ heavy duty rolls. 42″ × 72″ revolving 5/8 in. screen. No. 2 rolls 42″ × 14″. Screens. No. 3 rolls 42″ × 14″. 10 amalgamation plates, 2 mechanical classifiers. Sands to 7 reciprocating tables, slimes to 10 Frue vanners. Cyanide dept. Blaisdell equipment for sands. Slime agitation and Butter's filter. Built by Traylor Engineering Co., N. Y.

Cyanidation in Nevada: L. M. King. *M. and S. P.*, 96, 123. Gives data on tests on ores from the Goldfield Con. Mines Co. and outlines a method of economical treatment.

Treatment of Slimes in the Black Hills: S. E. Bennett. *Mining World*, 28, 323. Description of methods used at the Homestake, filter presses, Merrill precipitation presses, etc.

Some Features of Silver Ore Treatment in Mexico: W. A. Caldecott. *J. Chem. Met. and M. Soc. S. A.*, VIII., 203, 266. Unskilled labor is cheap and irregular. An electrolytic refinery for gold and silver bullion in Mexico City in 1906. Metric system used in dealing with Mexican silver ores. Cyanide consumption expressed in kilos per metric ton (2,204 lbs.). Silver in grams per metric ton. 1 gram per metric ton = 14 grains per 2,000 lbs. 1 kgm. per metric ton = 2 lbs. per 2,000 lb. ton. Peso or Mexican dollar is worth 25 pence or 0.5 dwt. gold, legal value is 0.75 gm. fine gold. Fluctuating value of silver a disturbing factor. Ore generally hard flinty quartz, containing little extraneous base metal mineral. Silver chiefly as sulphide. A small percentage of gold occurs. Essential features of treatment are fine crushing with cyanide solution and treating the bulk of the resultant pulp as a slime by decantation followed by mechanical filtering and zinc shavings precipitation of the solution. The second part of the paper deals with theoretical considerations.

Development of the Cyanide Process for Silver Ores in Mexico: B. MacDonald. *E. and M. J.*, 85, 802. Gives some notes and costs of the patio process. Pan amalgamation tried. Then the cyanide process was proved successful by E. M. Hamilton but not directly taken up. In 1905 the first cyanide plant went into operation and has now displaced the older methods.

Present Cyanide Practice in Mexico: M. R. Lamb. *E. and M. J.*, 85, 703. Most cyanide plants are treating silver ores. At Guanajuata more than 600 stamps are in operation. Pinguico Mill, 2 Blake crushers because ore is sticky, 40 stamps through 8-mesh. Two sizes by launder. Coarse to 2 Chili mills, 40-mesh. Joins undersize and is

classified by a "V" box 20 ft. long, 5 ft. high, sides sloping at 60°. Twelve Wilfley tables. Middlings to a Bryan mill and then reconcentrated on 4 Wilfleys. All slime and tails to 2 Dorr classifiers. Sand and slime. Sand plant. Slime goes to a cement table which separates a fine concentrate and slime goes to slime plant with vacuum filter.

Guanajuata Reduction and Mines Co. has 160 stamps: cone classifiers into sand and slime. Milling in cyanide solution. Abbe tube mills. Peregrina mill 100 stamps. Wilfleys. Tube mills. Blaisdell system. San Prospero mill does most of the custom work of the camp. 40 stamps, Wilfley tables, cement slime table, tube mill, sand and slime plant. Pastitas. 80-stamp mill of the Guanajuata Con. Co. treats high grade ore of the Serena mine. Wilfley and new Standard tables. Finer part then over Frue vanners. Slime agitation. Ridgway filter being installed. Jesus Maria mill of Guanajuata Amalgamated Co. near La Luz, 100 stamps, concentrators and cyanide sand and slime plant. The Brown agitating vats used at Pachuca by Grothe. Loretto mill has 40 stamps, Wilfleys, Chili mills for regrinding, Frue vanners and agitation for cyaniding the tailings.

Cyanidation of Silver Ores at Guanajuata: B. MacDonald. *E. and M. J.*, 85, 710. Describes work at the Hacienda de San Francisco fully with some excellent photographs and a plan of the mill.

The Dos Estrellas Mill. M. and S. P., 96, 197. The mines and two mills of the Compânia Minera Las Dos Estrellas are in the state of Michoacan but within 3 miles of El Oro. Ores were crushed in stamp-batteries, classified, reground in tube mills and cyanided. Mill No. 1 has 130 stamps, crushing in cyanide through 20–30 mesh. 4 tube-mills 24 ft. × 5 ft., El Oro ribbed lining cast from chrome steel and scrap iron. 75 per cent. of pulp passes 80-mesh. Cones. First sand wash 0.22 per cent. KCN, transferred. Second washes 0.4 per cent. followed by 0.12 per cent. Agitation of slime. Burt filter-presses being installed. Mill No. 2 has 120 stamps, 5 tube mills, Allis-Chalmers 24 × 5 ft. Cyanide plant has Blaisdell excavators and Robbins belt conveyors. Costs per ton. Milling 1.24: cyaniding 1.70. Mining 4.96. Others 1.28 = 9.18 Pesos.

Properties of the New York and Honduras Rosario Mining Co.: F. C. Nicholas. *Mining World*, 28, 367. It is proposed to do away with amalgamation (pan) and use cyanide process.

Waihi Ore Treatment: Notes on: R. Stokes. *J. Chem. Met. and Min. Soc. S. A.*, VIII., 10. The Waihi Co. has three mills, the Victoria, the Waihi and the Union, of 200, 90 and 40 stamps. Peculiar from its use of honey-comb tube mill liners, the independent placing of the amalgamating house, and the vacuum slimes plant.

The Waihi Gold Mine in New Zealand, I., II.: R. Stokes. *Mining World*, 28, 59, 103. Description of the district, deposits, milling notes, position of workmen, costs.

Metallurgy of the Kalgoorlie Goldfield: C. W. Williams. *E. and M. J.*, 85, 345. Two methods are used: (1) Wet crushing, classifying and concentrating plants with or without bromo-cyanide: (2) all-roasting, dry-crushing plants with auxiliary fine grinding to slimes and

filter-pressing. To class one belong the Ivanhoe, Golden Horseshoe, Oroya-Brownhill, Lake View Consols. The all-roasting plants are the Great Boulder, Perseverence, Kalgurlie, South Kalgurlie, Associated and Associated Northern. In general the ore is dry-crushed in Krupp or Griffin mills, roasted in Edwards or Merton furnaces and after fine crushing in grinding pans is agitated and filter-pressed. The paper gives numerous tables of products, work, costs, etc.

Westralian Wet-crushing Plants with Some Notes on Labor Efficiency: G. W. Williams. *J. Chem. Met. and M. Soc. S. A.*, VII., 232. Gives tables of battery work of the wet-crushing mill. Amalgamation at all wet-crushing plants. Where very coarse screen used, tables discarded and all amalgamating done in grinding pans. Concentration. Fine grinding, pans of Forwood-Down type. Final grinding to slimes (— 150 or — 200) in tube mills ("grit mills"). Agitation. Filter-pressing. Tables of expenses of Golden Horseshoe Estates Co., Ltd., and for labor distribution.

Electro-cyanide Processes: D. Lay. *E. and M. J.*, 85, 765. Discusses "electro-cyanide" processes which aim at precipitation of the precious metals during agitation of the finely-ground ore and cyanide solution.

The Clean-up, Melting and Refining of Gold Bullion: G. W. Williams. *Mining World*, 28, 21. Extract from *Monthly Journal of W. A. Chamber of Mines*, June 29, 1907.

A Small Lixiviation Plant in Mexico: H. A. Horsfall. *Mining World*, 28, 515. Ocampo mill on the Rio de Morris, S. W. Chihuahua. Ores from La Cruz mine. Originally used Kiss process. Reconstructed, 2 roasters for the stamped ore. Ore washed and leached with 0.75 per cent. hyposulphite of soda. Precipitation of polysulphide of lime. Silver sulphide run down with lead on cupel hearth. Four tables of work given.

Iron and Steel.

The Last of the Jersey Forges: E. P. Buffet. *E. and M. J.*, 85, 309. The bloomery forges made bar iron direct from the ore, the German Catalan method. In 1784 there were 79 forges and 8 furnaces. In 1802, 150 forges, and about 1830 there were 108 forge fires and 28 furnaces of which 12 were blast furnaces and the rest foundries. On the Rockaway and Pequanock rivers remains of the industry are found. Swedeland forge at Milton, Burnt Meadow forge, and Ryerson's forge at Bloomingdale.

New Austrian Iron Works: T. Naske. *Stahl und Eisen*, 1907, 1645. Description of new equipment of Witkowitzer Bergbau- und Eisenhüttengewerkschaft: Eisenwerk Trymietz de Oesterreichischen Berg- und Hüt. Eisenwerk Kladno der prager E. Ind. Ges.,

Iron Making in Australia: A. Selwyn-Brown. *E. and M. J.*, 85, 601. Describes work done by William Sandford at Lithgow, N. S. W., in his blast furnace and open hearth plant, which shut down for want of working capital.

Development of Electricity in the Iron and Steel Industries: D. Selby Bigge. *J. I. and S. Inst.*, 1907, I., 57. Special reference to application to reversing rolling mills of high power.

The Metallurgy of the Martin Process: T. Naske. *Stahl und Eisen*, 1907, 127. Study of the behavior of manganese, silicon, carbon and phosphorus in the open hearth furnace.

Method of Producing High-Class Steel from Pig Iron containing Chromium, Nickel and Cobalt: A. W. Richards. *J. I. and S. Inst.*, 1907, I., 114. Open hearth process produced steel with Ni 1.5 per cent., Co 0.25 per cent., Cr 0.30 per cent., of exceptionally good qualities.

The Duplex Process for Steel Making: H. M. Howe. *Electrochem. and Met. Ind.*, VI., 7. Removes the silicon and part of the carbon of pig iron in an acid converter and then removes phosphorus and brings the steel to the desired composition in a basic open hearth furnace. Reasons for the resulting composition of the steel.

Manufacture of Steel from High Silicon Phosphoric Pig Iron by the Basic Bessemer Process: A. W. Richards. *J. I. and S. Inst.*, 1907, I., 104. Iron oxide put in converter, molten gray Cleveland pig (low in S, Si = 1.5 to 3 per cent.) poured on. Blown till all Si is oxidized. Turned down and fluid slag poured off, etc.

The Induction Furnace and Its Use in the Steel Industry: v. Engelhardt. *Electrotech. Zeits.*, Vol. 28, Nos. 44–47. Abst. *Electrochem. and Met. Ind.*, VI., 143. Gives illustrations of various designs, relation between frequency and furnace capacity, and between kilowatts per ton and size of furnace.

Modification of the Induction Furnace for Steel Refining. Electrochem. and Met. Ind., VI., 10. Description from *Stahl und Eisen*, Nov. 6, 1907, by Professor Wedding of a new development by Roechling and Rodenhauser. The furnace is intended to refine molten steel received from the basic converter and to produce in continuous operation soft iron equivalent to Swedish.

Electric Furnaces: Pitwal. *C. R. de la Soc. de l' Ind. Min.*, March, 1907, 88. Engelhardt. *Stahl und Eisen*, 1907, 807. In operation there are 14 Kjellin, 10 Heroult, 2 Ginn, 3 Stassano, 2 Keller, 2 Frick, 1 Schneider, 1 Wallin, 1 Colby, 1 Girod.

Pig Iron from the Electric Furnace: B. Neumann. *Stahl und Eisen*, 1907, 1256. Discussion of results in making pig.

Iron Alloys and Metals for the Steel Industry: W. Venitor. *Stahl und Eisen*, 1908, 41, 22; 149, 225. Discusses Spiegel, ferro-manganese, ferro-silicon, silico-spiegel, Al, and ferro-aluminium, ferro-chrome, ferro-nickel, etc.

Melting Iron in the Foundry Cupola: H. McCormack. *Electroch. and Met. Ind.*, VI., 21. Work undertaken to obtain valuable data on cupola practice.

Ferro-Vanadium for Foundry Purposes. Electroch. and Met. Ind., VI., 127. From *Trans. Am. Foundrymen's Association.* Alloys con-

taining up to 0.45 per cent. Va were made. The strongest was 0.22 per cent. Va, 0.5 per cent. Si, 0.54 per cent. Mn and ran 77,050 lbs. per sq. inch.

The Uniform Nomenclature of Iron and Steel. Bull. A. I. M. E., 1908, 227. Report of Committee 24 Internat. Assoc. for Test-Materials. Gives names in English, French, German, Swedish, Danish and Dutch of iron, steel, etc., and the apparatus in which they are made. Then follows definitions, glossary of sizes, shapes, etc.

The Nomenclature of Iron and Steel. J. I. and S. Inst., 1907, III., 216.

The Relation between the Process of Manufacture and Some of the Physical Properties of Steel: F. W. Harbord. *J. I. and S. Inst.*, 1907, I., 181. Some 53 bars were tested for T. S., by Brinell's ball test for hardness and by different impact testing in the rolled condition and after heating to 620° C. In order of strength we have acid Bessemer, basic Bessemer, acid O. H. and basic O. H. lowest. Hardness is similar.

The Aging of Mild Steel: C. E. Stromeyer. *J. I. and S. Inst.*, 1907, I., 200. A number of failures were collected with details. Tests made to find whether any or all qualities of mild steel possess aging qualities and whether such aging qualities are the cause of failures in practice.

Further Experiments on the Aging of Mild Steel: C. E. Stromeyer. *J. I. and S. Inst.*, 1907, III., 86. Contains drop tests, Brinell tests, microstructure and bending tests made on the above material.

Carbon-Tungsten Steels: T. Swinden. *J. I. and S. Inst.*, 1907, I., 291. Nine steels W = 3 per cent., C = 0.144 to 1.24 per cent. were made. Were normalized by heating to 950° C. for 15 minutes and cooled in air. Tensile tests, crushing tests, alternating stress tests, bending tests. Pyrometric work: Microstructure. Fe_3W formed. 3 per cent. W raises tenacity without lessening ductility. Maximum tenacity with 0.9 per cent. C. Below a certain initial temperature all the steels examined show critical points as for carbon steels. Heating beyond this initial temperature lowers Ar_1 to 570° C. Structure resembles carbon steels but finer in pattern.

The Corrosion of Water-Jackets of Copper Blast-Furnaces: G. B. Lee. *Bull. A. I. M. E.*, 1908, 179. Describes corrosion of jackets and gives analyses of water used, which, however, did not corrode the boiler tubes. Discussion by several members.

Brinell Ball Press: E. Preus. *Baumaterialienkunde*, 1907, 295. Describes apparatus put out by the Alpha Co., Stockholm.

The Laboratory of a Modern Blast-Furnace Plant: A. Kayser. *Stahl und Eisen*, 1907, 1315, 1335. Description of plans and equipment of what an iron-works laboratory ought to be.

LEAD.

Lead: Its History and Economic Development, I.: E. W. Buskett. *Mining World*, 28, 447. Properties, occurrence, mineralogy, assay, metallurgy, desilverization, uses.

The Greenside Lead Mines, Cumberland: E. T. Borlase. *E. and M. J.*, 85, 297. Gives geology, mining methods, milling plant, smelting works. Reverberatory roaster, 16-cwt. charges. Roasted concentrates are smelted in water-back Scotch hearths. Lead runs 13 oz. Ag. Slags smelted in rectangular lead blast furnace. Flues ¾ mile long.

Nagybanya, Hungary: E. Skewes. *M. and S. P.*, 96, 66. Eastern part of Hungary bounded on the east by Roumania and Galicia. Thirteen hours from Buda-Pest. Short description of the mines, their output, etc. At Fernezely there is a government smelter. Several new reverberatory furnaces being installed. Siberian hearth seen at work. Blast furnace house built in 1853 now idle.

The Relations between Miners and Smelters. E. and M. J., 85, 222. Abstract from report of H. S. Joseph to American Mining Congress at Joplin, Mo., Nov., 1907. Gives prices in Salt Lake Valley. Gold $19.50 per oz. Silver, 95 per cent. of fine assay at New York price. Copper, all in excess of 0.5 unit (10 lbs. per ton) paid for at wholesale price of electrolytic in New York market less 2⅞ c. per lb. If in a lead ore 4.5 c. is deducted. Lead, 90 per cent. paid for at New York price of desilverized lead less 1 c. per lb. No pay for less than 3 units. Treatment charges. Initial charge $1 per ton for lead ores. 10 c. per unit of insoluble matter. 30 c. per unit of zinc in excess of 10 per cent. 25 c. per unit of sulphur above 2 per cent. Maximum sulphur penalty being $1 per ton. 25 c. per unit of Speiss in excess of 5 per cent. On iron a credit of 10 c. per unit is allowed.

Advocates organization of mine owners and operators throughout the country for the purpose of building, operating and maintaining their own mills and smelters.

The Price at which Profits Vanish: J. R. Finlay. *E. and M. J.*, 85, 165. The Cœur d'Alene mining companies sell their ore to the smelter who pays for 90 per cent. of the lead at 90 per cent. of the New York price or 81 per cent. of the full quantity and price when lead sells at 4.10 c. or under. When the price rises above 4.10 c. the smelter pays 81 per cent. on half the additional price. Payment is made on 95 per cent. of the full value of the silver. A freight and treatment charge of $16 per ton is deducted from the value of average concentrates.

Figures are given to show that a good deal of lead might be mined at 3 c. per lb., New York price.

The Dwight and Lloyd Sintering Process: A. S. Dwight. *E. and M. J.*, 85, 649. An illustrated article which points out that in ordinary pot-roasting the sintering action starts below and moves slowly up to top, and that particles are subjected to the maximum heat of sintering (1000–1200° C.) for only a minute or two. Hence the greater part of the capacity of the pot is for storing. The Dwight and Lloyd machine consists of a pair of circular rims of iron carrying a set of cast-iron herring-bone grates, the whole forming a drum-shaped structure acting like an endless conveyor. Inside the drum and occupying the top quadrant is a stationary suction box connected with a suction fan. A 4-inch layer of ore is fed upon the grated face of the drum and passes under the igniter

which kindles the top surface of the ore. Air is sucked down through the moving layer as it passes across the suction box. Twenty minutes is required to complete the action. Discharge automatic by a pointed grizzly frame, which strips the end of the finished sinter-cake from the face of the wheel like the bark from a tree.

Electrolytic Lead Refining and Treatment of Slimes and By-Products: A. G. Wolf. *Western Chemist and Met.*, III., 81. See "The Betts Process at Trail, B. C.," *M. and Mins.*, 28, 11.

NICKEL TIN.

Recovery of Nickel from Oxide and Silicate Ores: W. Koehler. *Electroch. and Met. Ind.*, VI., 145. Process to handle silicate ores, etc. Reviews methods already tried. Hydrometallurgical process consists of chloridizing (roast or other method), then oxidizing roast to convert unstable chlorides into oxides and saving chlorine, leaching out soluble magnesium chloride and then dissolving out the nickel with dilute HCl. Solution electrolyzed for nickel or nickel precipitated as oxide. Improvements in the process described.

Electric Melting of Tin Ores: F. Mattonet. *Metallurgie*, 5, 186. Ore ran 50 per cent. Sn, 7.6 per cent. Fe, 3.3 per cent. Zn, 11 per cent. SiO_2, etc. Roasted. Melt in Kryptol furnace with carbon and flux and obtained tin 96.3 per cent. and better with temperature 900–1000° C.

ZINC.

A Review of the Zinc Industry: H. M. Burkey. *Electroch. and Met. Ind.*, VI., 99. Review for 1907 on ores and their dressing, roasting, smelting, thermoelectric smelting, lixiviation, zinc-oxide, galvanizing. Bibliography of 12 titles.

The Wilfley Furnace: J. M. McClave. *E. and M. J.*, 85, 453. Describes a furnace suitable for roasting iron sulphides in preparation for magnetic separation. At works of Colorado Zinc Co., Denver, a roaster is used together with cooling apparatus, 4 Ding magnetic separators and 7 Wilfley tables. Plant capacity 100 tons a day.

Flotation Processes at Broken Hill. M. and S. P., 96, 494.

Minerals Separation Ltd. — Used on the Central Mine of the Sulphide Corporation. 2,000 tons a day. Produce a concentrate of 47 per cent. Zn and 10 per cent. Pb. 85 per cent. extraction. *Potter Process.* — Consolidation of Delprat and Potter interests. Broken Hill Proprietary using Potter process only. Tails from Wilfleys mixed with reground material from dumps, washed clean of slime, and fed to Potter pans 9 × 9 ft. at top 16 ft. deep where it comes in contact with strong acid solutions. Extractions of zinc 90 per cent. Plant treats 1,200 tons a day. *De Bavay Process.* — Used at Broken Hill North Mine. Tails from lead-mill washed and sized through 60-mesh. A. Z. Clark agitator, 15 tons a charge. Acid wash. Weak sulphurous to which is added 5 to 10 lbs. per ton of tails of strong sulphuric acid. Transferred to pressure vat and an oil emulsion added. Forced back into storage hoppers and thence fed in thin stream into inclined tables. Oiled and

aerated mineral floats off. Gangue sinks. *Elmore Vacuum Process.*— New plant of Zinc Corporation. 16 machines, 25 tons a day each. It is proposed to retreat the concentrate by roasting off the oil and then concentrating on Wilfleys to recover a portion of the lead. *Gillies Process.*— Has been abandoned on Block 10 mine because of high costs and low extraction.

The Dressing of Zinc Concentrates out of the Broken Hill Tailings: C. Göpner. *Metallurgie*, 5, 128. Review of different trials of various methods, Delprat, Potter, Cattermole, Elmore. The last proved successful.

The Elmore Vacuum Process: B. Haywood. *Western Chemist and Metallurgist*, IV., 100. The process depends on the affinity of oil for sulphide mineral in the presence of water and that of a bubble of gas for any kind of oil also in the presence of water. In a vacuum the number and lifting power of the bubbles greatly increase. The gas is generated in the pulp itself by decomposing the water by electric current or by sulphuric acid. The latter method is most convenient. 20 lbs. acid per ton of crude ore is the maximum used. Over 100 plants now in operation.

The Elmore Concentration: C. Göpner. *Metallurgie*, 5, 1. Description of process and results at Glasdir Copper Mine, North Wales. Used at Lake View Consols, Kalgurlie, W. A. Table of ores and products. See: P. Linde, *Metallurgie*, 5, 87.

Fuels, Refractories, etc.

Government Fuel Testing Plant: G. R. Delamater. *M. and Mins.*, 28, 401. The Technologic Branch of the United States Geological Survey recently established a plant in Denver, Colo., to test western coals. A coal washery and two beehive coke-ovens.

The Calorific Value of Coal as Determined in the Mahler Bomb Calorimeter: G. A. Burrell. *Western Chem. and Met.*, IV., 6. Gives the principle of the apparatus. One determination takes 35–40 minutes. Coal 60 mesh, briquetted. Oxygen, pure, at 18 atoms pressure. Waterjacket of about 4 liters. Thermometer read to 0.001°. Determination of water equivalent. Method of making a determination. Correction for radiation, etc.

The By-Product Coke Oven in the United States: C. G. Atwater. *M. and M.*, 28, 317. Gives table showing location, number of ovens, use of coke and use of surplus gas — 13 of United-Otto, 13 of Semet-Solvay, 2 Rothberg and 1 Koppers plant. Analysis of growth of by-product oven.

The Von Bauer Coke-Oven System. Electroch. and Met. Ind., VI., 108. From *Stahl und Eisen*, No. 24, 1906, by O. Simmerbach. Description with many illustrations. Plant being introduced into the United States.

Refractories: J. R. Campbell. *M. and Mins.*, 28, 457. Materials used in construction of coke ovens. Firebrick, loam, and clay filling. Analyses.

The Utilization of Blast Furnace Gases for Power Production: A. L. O. Genter. *Western Chem. and Met.*, III., 99. Discusses use in gas engines, calculation of thermal value of gases, washing of gases, etc.

Purification of Gases: O. Nagel. *Electroch. and Met. Ind.*, VI., 102. Treats of methods and apparatus for separation of gases from solids, from liquids and from other gases.

GENERAL.

Old Fashioned Metallurgy: F. W. Traphagen. *Western Chem. and Met.*, IV., 131. Extract from an old book in 4 parts containing a translation by the Earl of Sandwich in 1669 of Albaro Alonzo Barbas' Art of Metals.

Metallurgical Calculations: J. W. Richards. *Electroch. and Met. Ind.*, VI., 8. The metallurgy of lead. Reduction of lead oxide, oxidation of sulphate, double reactions, oxidation refining. The volatility of lead.

VI., 63. Roasting of lead ores, reduction of roasted ore.

VI., 112. The metallurgy of gold and silver, electrolytic refining. Volatization of silver and gold.

VI., 141. Metallurgy of zinc, roasting of blende.

Melting, Heating, Dissolving and Eluting: O. Nagel. *Electroch. and Met. Ind.*, VI., 68. Description of various apparatus such as the reverberatory furnace, cylinders, digesters, dissolving apparatus.

Drying apparatus: O. Nagel. *Electroch. and Met. Ind.*, VI., 147.

On the Direct Separation of Iron from Pyrite by Aluminium Powder and the Probable Formation of a Double Sulphide, $Al_2S_3.FeS$: H. Ditz. *Metallurgie*, 4, 786. With FeS_2 and FeS, aluminium only reduces part of the iron, the rest is found as FeS probably combined as $Al_2S_3.FeS$.

The Affinity of Sulphur for Different Metals: E. Schütz. *Metallurgie*, 4, 659, 694. Melts of different metals and sulphides were made two at a time and the resulting matte and bottoms analyzed. Mn + FeS gave a matte rich in Mn and an iron bottom. 99 per cent. of sulphur goes to Mn matte. Similar results from melting MnS + Fe. In the case of Cu_2S and Fe or FeS and Cu, the greatest part of the sulphur stays with the copper. For Sn and FeS or SnS and Fe the sulphur has a greater affinity for the iron. For Ni and FeS or Ni_2S and Fe, nickel has the greater affinity. For Ni and ZnS or Ni_2S and Zn, nickel combines with the sulphur and zinc is distilled. For zinc and PbS, sulphur has greater affinity for the zinc. The series runs Mn, Cu, Ni, Fe, Sn, Zn, Pb as regards affinity for sulphur. CaS and BaS worked with.

Some Determinations of the Actual Temperatures of Cupellation and Certain Other Interesting Phenomena in the Cupellation of Silver: C. H. Fulton. *Western Chem. and Met.*, IV., 31. Difficulties of work. Quartz and iron tubes corrode and spoil thermocouples. Iron pipe covered with chromite finally stood the corrosion of the lithage. First a series of determinations were made on the cupelling lead and the muffle near it. Maximum difference 245° C., lead being hotter due to oxidation. The lead "froze" at 802° C. the muffle being 675° C. Buttons

uncovered at 800° and 804° C. and "froze" at 804° and 788° C.—a second lot gave 832°, 829° and 834° "uncovering" and 850° C. freezing and so on. Recent research has given the melting point of pure lithage at 906° and 884° C., and during the soft pasty stage there is marked volatilization of PbO (800° C.). Thus the uncovering is far below the melting of pure lithage. But 6PbO, SiO_2, 5PbO, SiO_2 and 4PbO, SiO_2 are thinly fluid at 794°, 796° and 726° C., and small traces of siliceous slag would form such a thin film of silicate and materially lower the "opening" temperature. Silver beads finishing with suffusion are free from lead. "Feathers" indicate proper temperature provided air draft is not excessive.

Lead Oxide and Silica: W. Mostowitsch. *Metallurgie*, 4, 647. Mixtures from 95.68 to 78.68 per cent. PbO or 6 PbO.SiO_2 to PbO.SiO_2 were melted, temperatures of formation and cooling curves taken. PbO and SiO_2 form silicates between 700° and 800° C. Solutions of PbO in lead silicate.

Decorative Etching on Silverware: A. F. Saunders. *Met. Ind.*, VI., 120. The design is painted over with a ground of asphalt 3, beeswax 1 in turpentine. Dip in nitric acid (20 per cent.) 10 to 15 minutes.

The Mechanical Equipment of the Ottawa Mint: A. H. W. Cleave. *Can. Min. J.*, 29, 100, 174, 203. An interesting description of the installation of the new Ottawa Mint. A refinery will shortly be erected and gold treated electrolytically.

Preparation of Materials for the Blast Furnace: D. Baker. *E. and M. J.*, 85, 609. Discusses the subject of sulphur and the cost of smelting, form of impurities, concentrates, nodulizing processes, effect upon reduction, etc.

The Smelter Smoke Problem: F. W. Traphagen. *Western Chem. and Met.*, III., 46. Discusses damage done to soil, crops and live stock.

Feeding Experiments with Forage Grown in Smelter Zone: L. A. Merrill. *Western Chem. and Met.*, III., 59. Work carried out in Salt Lake district.

Alloys.

How to Make a New Alloy: E. A. Lewis. *Met. Ind.*, VI., 1. An analysis of brass is given and its method of manufacture reasoned out.

Standard Government Alloys. Met. Ind., VI., 2.

Mark.	Cu	Sn	Zn	T. S.	E. L.	El. 2"	R. A.	Uses.
A	84	16	2	17,820	17,820	Nil.	Nil.	Bearings, bushings, sockets.
B	86	14	2	17,020	17,020	"	"	Guides, valves and plug cocks.
C	88	10	2	30,360	17,190	20	9.8	Torpedo tubes, general fittings.
D	90	10	2	22,610	15,400	15.5	11.7	Bolts, nuts, stanchions.
E	67	—	33	26,770	7,760	50	39	General fittings, drain pipes.
F	—	—	—	Scrap.	—	—	—	Deck and name plates.
G	77	28		27,180	27,180	Nil.	Nil.	Ship bells.
H	33	—	67	—	—	—	—	Electrochemical quality of iron.
I	60	—	40	47,170	15,110	54	46.7	Muntz metal.

New Bearing Metal: L. Semple. *Metallurgie*, 4, 667. The alloy 91.7 per cent. Al and 7.61 per cent. Cu is compared with the tin bearing metal Sn 83.33, Sb 11.11, Cu 5.55 per cent.

New Metal Alloy. Met. Ind., VI., 2. Patent by E. Haynes, luster, malleability high, non-corrosive. Nickel with 30 to 60 per cent. chromium. Made in magnesia crucible by melting metals or reduction of oxides by aluminium.

The New Alloys. London Mining Journal, Sept. 21, 1907, p. 361. From *La Nature.* Abs. *J. Chem. Met. and M. Soc. S. A.*, VIII., 159.

Cupro-Magnesium. — Cu 90, Mg 10. A deoxidizer.

Phono-electric Wires. — Cu 98.55, Sn 1.4, Si 0.05. For telephones and trolleys. Electric conductivity two-fifths that of pure copper.

Sterline. — Cu 68.5, Zn 12.84, Ni 17.88, Fe 0.76. White metal. Substitute for silver.

Manganese Resistance Metal. — Cu 85, Fe 3, Mn 12. A substitute for german silver in resistance boxes. Specific resistance 3/100 to 4.5/100 that of copper.

Manganine. — Cu 82.12, Ni 2.29, Fe 0.57, Mn 15.02. High fusing point, low temperature coefficient, for resistance boxes.

Acid Proof Metal. — Cu 82, Zn 2, Sn 8, Pb 8. Used in paper works where the bisulphite process is employed. Only proof against weak acids, except nitric, which attacks it quickly.

Victor Metal. — Cu 49.94, Zn 34.27, Ni 15.4, Al 0.11, Fe 0.28. Used for german silver. Not attacked by salt water.

Aluminium Silver. — Cu 57, Zn 20, Ni 20, Al 3. White. Very tenacious. Remains bright in air.

Tempered Lead. — Pb 98.5, Sb 0.11, Sn 0.08, Na 1.3. Can be rolled. Used for manufacture of shaft bearings.

Alkali Proof Metal is iron with 5 per cent. to 10 per cent. nickel. All alloys containing zinc, tin, lead, aluminium, antimony or silicon are readily attacked by caustic alkalies.

Annealing Brass in Rolling: A. W. L. *Met. Ind.*, VI., 55. Gives the mode of handling a bar of metal 6 in. wide × 1⅛ in. thick, as to passes, reduction and annealing.

Distribution of Rolling Mills. Met. Ind., VI., 126. Chief brass center is Waterbury, Conn. Plants at New York, Hastings, Rome and Buffalo, Detroit, Kenosha, Wis. Copper mills at New Bedford, Taunton, Belleville, N. J., Baltimore, Pittsburgh and Lisbon, O.

PYROMETRY — METALLOGRAPHY.

Principles of Microscopic Metallography. L. Daunis. *Revista Tech.-Ind.*, March, 1908, 77. Deals with preparation of specimens, methods of examination with several examples, photographs of cast iron, wrought iron, steel, brass, etc.

Sentinel Pyrometers and their Application to the Annealing, Hardening and General Heat Treatment of Tool Steel: Brearley and Moor-

wood. *J. I. and S. Inst.*, 1907, I., 261. Propose the use of salts of metallic oxides. NaCl melts at 770–775° C., KCl at 740° C., $BaCl_2$ at 930° C. and so on. Curve for mixtures of NaCl and KCl given. Lowest point (eutectic) = 655° C. Also curves for Na_2So_4 and K_2SO_4, for Na_2CO_3 and K_2CO_3, for K_2CO_3 and KCl, etc.

Some New Types of Electric Furnaces for Laboratories: K. Friedrich. *Metallurgie*, 4, 778. Three types of handy resistance furnaces, one tube and two for crucibles.

A Graphical Representation of the Solidification of Eutectic Alloys: A. Sauveur. *Electroch. and Met. Ind.*, VI., 18. Illustration of solidification of an alloy composed of excess metal and eutectic when solid.

On the Structure of Metals and the Changes due to Working and Annealing: W. Campbell. *Metallurgie*, 4, 801, 825. The structure and changes due to work and heat treatment of Al, Sb, Bi, Cd, Cu, Au, Pt, Pb, Ag, Sn and Zn are shown.

The Binary and Ternary Alloys of Tin, Lead, Bismuth and Cadmium: A. Stoffel. *Zeits. anorg. Chem.*, 53, 137. The binary system, lead and tin. Work of Charpy, Kapp, Wiesengrund, Stoffel and Heycock and Neville given and temperatures plotted. The eutectic freezes at 183–184° C., 72–76 at per cent. Sn.

The tin-cadmium series has a eutectic at 177° C. and 70.59 at per cent. Sn, whilst that of the tin-bismuth series occurs at 139° C. and 43 at per cent. Bi. The cadmium-bismuth eutectic is 44.5 at per cent. Bi at 146° C. That of bismuth and lead at 125° C. and 56.5 at per cent. Bi. Lead and cadmium have a eutectic at 67 at per cent. Cd and 149° C.

The tin, cadmium, lead diagram is given. Eutectic 145° C., 49.8 per cent. Sn, 31.96 per cent. Pb, 18.11 per cent. Cd (or 57, 21, and 21.9 at per cent.).

The tin, cadmium, bismuth diagram shows a eutectic at 34 at per cent. Sn, 27 per cent. Cd and 39 at per cent. Bi at 103° C.

Some of the alloys show evolutions of heat below the Pb eutectic point. Sn and Cd at 125° C. (Sn_4Cd). SnCd at 118–112° C. SnCdBi at 100° C. and 78–68° C.

Electrolytic Corrosion of Copper-Aluminium Alloys: W. S. Rowland. *J. Phys. Chem.*, XII., 180. The corroding effects of the more common sodium salts and of sodium hydroxide determined. The phases which have the greatest resistance to corrosion are those which have the poorer physical properties.

Aluminium and Copper: Carpenter and Edwards. *VIII. Report to the Alloys Research Committee. Inst. Mech. Engineers (London)*, Jan., Feb., Mar., 1907. Works out the properties of the copper rich and aluminium rich alloys, the microstructure and constitution of the series. Al_2Cu. 54 per cent. Cu 590° C. and Al form a eutectic at 33.5 per cent. Cu and 540° C. Between Al_2Cu and Cu a series of solid solutions form with transformations in the solid and reaction points.

The Alloys of Aluminium with Copper, Iron, Nickel, Cobalt, Lead and Cadmium: A. G. C. Gwyer. *Zeits. anorg. Chem.*, 57, 113. Copper-

aluminium curve consists of a minimum *B* at 88.5 per cent. Cu, a maximum *C* at 87.6 per cent. Cu (1038° and 1050° C.), two reaction points *D* and *E* at 56 and 45 per cent. Cu (625° and 590° C.) and minimum at the eutectic 32.5 per cent. Cu (544° C.). The series consists of (1) solid solutions up to 91.5 per cent. Cu or a (2) solid solution *B* or 88.5 per cent. Cu; (3) Cu_3Al, 87.6 per cent. Cu at *C;* (4) solid solutions of CuAl in Cu_3Al; (5) CuAl which forms by reaction of (4) with the melt *D;* (6) $CuAl_2$ which forms by reaction of CuAl with the melt *E;* (7) Al (3–5 per cent. Cu in solution) which forms a eutectic with $CuAl_2$ at 32.5 per cent. Cu and 544° C.

Iron and aluminium form a series whose freezing point curve consists of two branches which start from the freezing points of the two metals and meet at 50 per cent. and 1152° C. From 0 to 33 per cent. Al the alloys consist of solid solutions. From 58.2 to 100 per cent. Al, $FeAl_3$ occurs in a ground mass of Al. Between 66 and 48 per cent. Fe the alloys show lower points (1087°) and a eutectic structure. Between 47.5 and 40.8 per cent. Fe we find homogeneous polyhedra (solid solution).

Nickel-aluminium. Three compounds, NiAl 1640° C. $NiAl_3$ formed by reaction of NiAl with the melt 42 per cent. Ni (1135° C.) $NiAl_3$ by reaction of NiAl with the melt 27 per cent. (840° C.). $NiAl_3$ and Al form a eutectic at 6 per cent. Ni and 555° C. Between Ni and NiAl 68.4 per cent. Ni and 1640° C., we have a series of solid solutions with a minimum at 87 per cent. Ni and 1371° C. from which point a short horizontal runs to 80 per cent. Ni. This divides the series into 3 groups (1) solid solutions up to 87 per cent. Ni (2) solid solution 87 per cent. and solid solution 80 per cent. Ni (3) solid solutions of Ni and NiAl from 68.4 to 80 per cent. Ni.

Cobalt-aluminium show a similar curve. Max. at CoAl, 68.5 per cent. Co, 1628° C. Co_2Al_5 and Co_3Al_{13} formed by reaction with the melts at 38 per cent. 1170 and at 20 per cent. and 940° C. Below 20 per cent. Co we have Co_3Al_{13} in a ground mass at pure Al. Between CoAl and Co 1491° C., we have a series of solid solutions with a minimum at 90 per cent. Co 1375° C.

Lead and aluminium do not alloy, but show two freezing points, Al = 654° C. and Pb 327° C. Similarly cadmium and aluminium.

The Alloys of Calcium with Zinc, Cadmium, Aluminium, Thallium, Lead, Tin, Bismuth, Antimony and Copper: L. Donski. *Zeits. anorg. Chem.*, 57, 185. Three compounds of calcium and zinc, $CaZn_{10}$, $CaZn_4$, Ca_2Zn_3. Calcium and Ca_2Zn_3 (686° C.) form a eutectic at 57 per cent. Ca and 415° C. Ca_2Zn_3 and $CaZn_4$ form a eutectic at 17 per cent. Ca and 633° C. $CaZn_{10}$ shows a max. 717° C. and from this to the eutectic 17 per cent. Ca and 633° C. the curve falls, with a point at 13.3 per cent. (680° C.) where $CaZn_4$ forms. From $CaZn_{10}$ to pure Zn the alloys consist of crystals of the compound set in a ground mass of pure zinc. Lower points show the formation of CaZn (?) and Ca_4Zn (389° C.).

Cadmium and calcium form two liquid phases between 12 and 64 per cent. Ca. CaCd freezing at 685° C. Ca and CaCd form a eutectic at 72 per cent. Cd and 415° C. The Ca holds 13 per cent. Cd in solid

solution. $CaCd_3$ forms at 615° C. $CaCd_4$ and Cd form a eutectic at 1 per cent. Ca 8–10° C. below the melting point of cadmium.

Calcium and aluminium form two liquid phases between 16 and 45 per cent. Ca, $CaAl_2$ freezing at 692° C. $CaAl_2$ and Al form a eutectic at 8.1 per cent. Ca and 610° C. $CaAl_2$ and Ca form a eutectic at 549° C. and 74 per cent. Ca.

Calcium and thallium form two compounds CaTl and $CaTl_3$ (558 and 527° C.) and a solid solution 0.6 per cent. Ca freezing at 308° C. 6° higher than the freezing point of Tl. Alloys up to 15 per cent. made.

Calcium and lead form a compound $CaPb_3$ at a max. 649° C. And up to 6 per cent. Ca the alloys consist of crystals of this compound in pure lead. A second eutectic of $CaPb_3$ freezes at 623° C. and 9.7 per cent. Ca. Alloys up to 12 per cent. made.

Calcium and tin form a similar curve up to about 20 per cent. Ca. $CaSn_3$ freezes at 624° C., the eutectic at 603° C. and 15 per cent. Ca. Alloys up to 18 per cent. were made.

Calcium and bismuth form a eutectic at 260–265° C. with about 0.3 per cent. Ca. Above 4.5 per cent. Ca the alloys show an upper freezing point at 500° C. Alloys up to 10 per cent. made.

Calcium and antimony form a eutectic at 584° C. and about 8 per cent Ca. Alloys up to 9.1 per cent. Ca were made.

With calcium and copper 1 per cent. Ca changes the freezing point of of Cu 8° C. and 5 per cent. 74° C., while a eutectic point occurs at 920° C.

The Alloys of Copper with Cobalt, Iron, Manganese and Magnesium: R. Sahmen. *Zeits. Anorg. Chem.*, 57, 1. Copper and Cobalt (1493° C.) form two series of solid solution, the copper-rich with 5 per cent. cobalt and the cobalt-rich with 10 per cent. copper. The cobalt-rich reacts with the liquid to form the copper-rich "mischkrystal" at 1110° C. The former changes in the solid between 1110° and 1050° C., the latter at 950° C.

Iron 1532° and copper form two series of solid solutions. Iron-rich 3 per cent. copper, copper rich 3 per cent. iron, which is formed by the reaction of the former with the liquid at 1100° C. Change from gamma to beta iron at just below 800° C.

Copper and manganese (1214° C.) form a series of solid solutions with a minimum at 65 per cent. Cu and 866° C.

Magnesium (650° C.) and copper form two compounds only. Cu_2Mg at 800° C. and $CuMg_2$ at 570° C., Cu and Cu_2Mg form a eutectic at 10 per cent. Mg and 728° C. Cu_2Mg and $CuMg_2$ form a eutectic at 555° C. and 33 per cent. Mg; whilst $CuMg_2$ and Mg form a eutectic at 68 per cent. Mg and 485° C.

Copper-Nickel Alloys: Guertler and Tammann. *Zeits. Anorg. Chem.*, 52 (1907), 25. Form a series of solid solutions, the freezing point curve joining the freezing points of the two metals. The magnetic change falls from 320° in Ni to 115° C. in the 32 per cent. Cu alloy.

Silicon-Copper: E. Rudolfi. *Zeits. Anorg. Chem.*, 53, 217. Form a compound Cu_3Si (862° C.). This latter forms a eutectic with

Si (1404° C.) at 800° C. and 17.6 per cent. Si. Between 0 and 7.83 per cent. Si primary dendrites (*a*) containing up to 4.5 per cent. Si in solid solution freeze out at 849° C. These react with liquid 7.83 per cent. Si to form a second solid solution (*b*) which with Cu_3Si forms a eutectic at 820° C. and 9.8 per cent. Si. This second solid solution (*b*) breaks up into $Cu_{19}Si_4$ + (*a*) at 710° C.

Copper and Phosphorus: Heyn and Bauer. *Zeits. anorg. Chem.*, 52, 129. From Mitteilungen aus dem Kgl. Materialsprüfungsamt Gross-Lichtenfelde, 1906, p. 93. See *Metallurgie*, 1907, 242.

Alloys of Gold and Tellurium: T. K. Rose. *Bull. 41. I. M. and Met.*, Feb. 13, 1908. Form a compound $AuTe_2$ freezing at 452° C. $AuTe_2$ and Te (440° C.) form a eutectic at 20 per cent. Au and 397° C. $AuTe_2$ and Au form a eutectic at 60 per cent. Au and 432° C.

The Alloys of Iron with Tin and Gold: Isaac and Tammann. *Zeits. anorg Chem.*, 53, 281. Tin and iron form two liquid phases between 50 and 90 per cent. 0–19 per cent. Sn, solid solutions. At 1140° C. a solid *x* forms or freezes, has three forms, α, β, and γ, 496, 780° and 893° C. At 45 per cent. Sn the groundmass makes its appearance, freezing at 228° C.

Gold and iron form two series of solid solutions. At 1168° C., the critical point between the solid 28 per cent. Au, the liquid 74 per cent. Au and the solid 63 per cent. Au. Changes in the solid.

Alloys of Lead with Thallium and Indium: Kurnakow and Puschin. *Zeits. anorg. Chem.*, 52, 430. The curve for PbTl shows a max. at 380° C° (between Tl_2Pb and Tl_3Pb_2). From this point to pure Pb we have a series of solid solutions, while towards the Tl end of the curve we get a reaction point at 310.4° and 5.5 at per cent. Pb. Here the solid solution 24.7 reacts with liquid 5.5 at per cent. Pb to form solid solution 6.5 at per cent. Pb. Thus the series is represented by two series of solid solutions: (1) 0–6.5 at per cent. Pb, (2) 24.7 to 100 at per cent. Pb. Pure Tl freezes at 301° C. and transforms at 227° C.

The alloys of Pb and In consist of a series of solid solutions, the freezing point curve joining those of the two metals, the slope at the indium end being very gradual up to 10 at per cent. Pure In = 154° C. 6.93 at per cent. Pb = 154.7° C.

Indium and thallium also form a series of solid solutions.

Lead Thallium Alloys: K. Lewkonja. *Zeits. anorg. Chem.*, 52, 452. A summit is found at $PbTl_2$ about 374° C. Between this and Pb a series of solid solutions exist. From $PbTl_2$ and Tl 299.40° C., two curves intersect at 5 at per cent. Pb 309.5° C. where the solid solution of Tl in $PbTl_2$ (24 per cent. Pb) reacts with the liquid 5 per cent. Pb to form the solid 6.5 at per cent. Pb.

Palladium-Lead Alloys: R. Ruer. *Zeits. anorg. Chem.*, 53, 345. Form 5 compounds. Two occur at maxima, Pd_3Pb and 1219° C. Pb_2Pd 454° C., whilst $PbPd_2$ forms by reaction with the melt 43 per cent. Pd at 830°, and PbPd with the melt 27 per cent. Pd at 495° C., whilst the third forms by reaction with the melt 35 per cent. Pd at 596° C. A eutectic at 5 per cent. Pd 265° C. and minimum at 65 per cent. Pd and 1200° C. from which point the curve rises to Pd at 1541° C.

Mercury with Cæsium and Rubidium: Kurnakow and Żukowsky. *Zeits. anorg. Chem.*, 52, 416. Freezing point diagram worked out. $CsHg_2$, $CsHg_4$ and $CsHg_6$ shown as maxima. Other compounds indicated. 3 marked eutectics. 1.57 at per cent. Hg at 16.3° C. 75.75 at per cent. Hg 139.5° C. and 97.75 at per cent. Hg at — 46.6° C.

For rubidium the amalgams were made with 3.3 to 14.6 at per cent. Rb.

Alloys of Manganese with Copper and Nickel: S. Żemczużny, G. Urasow and A. Rykowskow. *Zeits. anorg. Chem.*, 57, 253. Manganese and copper form two series of solid solutions with a minimum at 34 at per cent. Mn and 868° C.

Manganese and nickel from a similar series with the minimum at 57 at per cent. Mn and 1030° C.

Phosphorus and Manganese: S. Żemczużny and N. Efremow. *Zeits. anorg. Chem.*, 57, 241. Manganese 1260° C. and Mn_5P_2 1390° C. form a eutectic at 9.5 at per cent. P and 964° C. Mn_5P_2 and MnP 1190° C. form a eutectic at 40.5 at per cent. P and 1095° C.

The Alloys of Bismuth and Nickel: Portevin. *Rev. de Met.*, 5, 110. Curve shows a gentle slope from Ni 1451° C. to 25 per cent. Ni and then becomes steep to the m. p. of Bi at 269° C. At 656° C. and 462° C. and 269° C. we have horizontals. The first shows reaction of Ni and melt 12 per cent. Ni to form NiBi (?). The second shows formation of $NiBi_3$, the last shows the freezing of the groundmass of pure Bi.

Nickel-Tin, Nickel-Lead, Nickel-Thallium, Nickel-Bismuth, Nickel-Chrome, Nickel-Magnesium, Nickel-Zinc and Nickel-Cadmium Alloys: S. Voss. *Zeits. anorg. Chem.*, 57, 34. Nickel and tin (1451° and 231° C.) form 3 compounds — Ni_3Sn_2, Ni_3Sn and Ni_4Sn. Between 3.5 and 18 per cent. Ni, also between 26 and 45 per cent. Ni, the melts exists as two liquid phases. Ni (with 15 per cent. Sn in solid solution) and Ni_3Sn form a eutectic at 1135° C. and 68.5 per cent. Ni.

At a lower temperature Ni_3Sn_2 breaks down into Ni_4Sn and Ni_3Sn_2. Ni_3Sn_2 (1264° C.) reacts with the melt at 1162° C. to form Ni_3Sn. At 42 per cent. Ni the eutectic of Ni_3Sn_2 and Sn (229° C. and 1.3 per cent. Ni) comes in. In the alloys of 60 per cent. and above Ni magnetism begins at 115–325° C.

Nickel-lead alloys exist as two conjugate liquids between 16 and 72° Ni, critical curve cutting freezing curve at 1338° C. With less than 16 per cent. Ni crystals of Ni occur in a ground mass of lead. The Ni holds 2 per cent. Pb in solution and has a magnetic change at 335° C.

Nickel and thallium (302°) form conjugate solutions between 0 and 90 per cent. Ni. The solid solution which freezes at 1385° contains 3 per cent. Tl. There is a magnetic change at 330° C. whilst the thallium transforms at 234° C.

Nickel and bismuth form two compounds, NiBi and $NiBi_3$. NiBi forms by reaction of Ni with the liquid 32 at per cent. Ni at 638° C. Whilst at 437 the compound $NiBi_3$ forms by reaction of NiBi with the melt containing 11 at per cent. Ni. The eutectic or ground mass is pure bismuth.

The nickel-chrome alloys have a peculiar curve consisting of two inclined branches from Cr 1555° and Ni 1452° which meet at 42 per cent. Ni and 1290° C. This point divides the series into two; solid solutions rich in Cr and those rich in nickel.

Nickel and magnesium form two compounds, Ni_2Mg and $NiMg_2$. Ni_2Mg, 1145° C., and Ni form a eutectic at 1084° C. and 10 per cent. Mg. Ni_2Mg reacts with the liquid alloy 46 per cent. Ni at 768° C. to form $NiMg_2$. $NiMg_2$ and Mg form a eutectic at 512° C. and 34 per cent Ni. Magnetic changes in the solid.

The nickel-zinc alloys were investigated up to 21 per cent. Ni. The alloys consist of crystals of $NiZn_8$ with zinc in solid solution (S.S = 14.5 per cent. Ni) set in a ground mass of zinc.

Nickel-cadmium alloys up to 15 per cent. Cd. Between 15 and 10 per cent. Ni primary *X* freezes, and at 501° C. reacts with the melt 10 per cent. Ni to form $NiCd_4$. Between $NiCd_4$ and pure Cd we find crystals of the former in a ground mass of the latter.

The Alloys of Silver: Portevin. *Rev. de Met.*, 5, 145, 182. Translation of work of Tammann and his pupils. AgAl, Petrenko; AgSb, Petrenko; AgBi, Petrenko; AgSn, Petrenko; AgMg, Zemczuznyj; AgNi, AgFe, AgCo, Petrenko; AgPd, Ruer; AgPb, Petrenko; AgTh, Petrenko; AgZn, Petrenko; AgPb, Doerinckel.

AuBi, Vogel; AuCd, Vogel; AuCu, Kurnakow and Zemcyuznyj; AuFe, Tammann and Isaac; AuNi, Levin; AuPd, Ruer; AuPt, Doerinckel; AuPb, Vogel; AuSb, Vogel; AuSn, Vogel; AuTh, Levin; AuZn, Vogel — from *Zeit. anorg. Chem.*

The Alloys of Silver with Lead and Tin: G. J. Petrenko. *Zeits. anorg. Chem.*, 53, 200. Silver and lead form a eutectic 97.5 per cent. Pb 303.3° C. Silver and tin show two sloping branches intersecting at the eutectic point 96.5 per cent. Sn 202° C., and a horizontal form 19 to 53 per cent. Sn at 48° C., showing the formation of Ag_3Sn by reaction of solid solution 19 per cent. Sn with liquid 53 per cent. Sn. A lower point at 252° C. is found between 19 and 53 per cent. Sn, due to a change in the Ag_3Sn.

Alloys of Silver with the Metals of the Iron Group (Fe, Ni, and Co): G. J. Petrenko. *Zeits. anorg. Chem.*, 53, 212. Ni and Ag form two liquid phases between 10 per cent and 96 per cent. Ni; nickel freezing from the alloys at 1465° C. Ni 1484° C. to Ni 4 per cent. Ag form a series of homogeneous solid solutions; below this point the ground mass of pure silver (961.5° C.) comes in. Silver and iron and silver and cobalt do not alloy.

Zinc-Thallium and Zinc-Iron Alloys: A. Vegesack. *Zeits. anorg. Chem.*, 52, 31. Form two liquid phases up to 95 per cent. Tl, the zinc freezing at 416° C. At 291° C. the eutectic freezes containing 3 per cent. Zn.

Zinc and iron, two compounds, $FeZn_8$ and $FeZn_7$. Iron added to zinc raises the freezing point 3° C., a solid solution being formed. With more iron crystals of $FeZn_7$ holding zinc in solution crystallize out. From 7.5 to 11 per cent. the alloys consist of $FeZn_7$ with Zn in

solid solution. Above this we have mixtures of $FeZn_7$ and $FeZn_8$. With the higher percentages an unknown crystallizes out about 800° C. but at 777° C., reacts with the melt (11 per cent.) to form $FeZn_8$. Similarly $FeZn_8$ reacts with the melt (4 per cent. Fe) to form $FeZn_7$ at 662°.

Zinc and Nickel: V. Tafel. *Metallurgie*, 4, 781. The freezing point diagram and microstructure, etc., worked out up to 50½ per cent. Ni. A compound $NiZn_8$ 876° C. and 23 per cent. Ni occurs. This holds some zinc in solution and the ground mass only appears with less than 12.2 per cent. Ni. Pure zinc and $NiZn_8$ form a eutectic at 418.5° C. and a very small percentage of Ni, so this ground mass is nearly pure zinc. At 45.8 per cent. Ni the alloy is again homogeneous and freezes at 1038° C. This body forms a eutectic with $NiZn_8$ at 27.5 per cent. Ni and 870° C. This eutectic disappears at 39.7 per cent. Ni. At a lower temperature, 640° C., a change in the solid occurs. From 45.8 per cent. onward a third body occurs in a groundmass of the second, with a change in the solid at 780° C.

The Alloys of Potassium with Aluminium, Magnesium, Zinc, Cadmium, Bismuth, Tin and Lead: D. P. Smith. *Zeits. anorg. Chem.*, 56, 109. Aluminium and potassium 657° C. and 63° C. are insoluble in the liquid and the solid state. So are magnesium (650° C.) and potassium.

Zinc and potassium form a compound KZn_{12} (585° C.) which in the solid changes from the instable to the stable form. KZn_{12} does not alloy with either zinc or potassium.

Cadmium and potassium form a compound KCd_{11} (487° C.). Between KCd (2.5 per cent. K) and Cd (322°) the alloys consist of crystals of the compound in a ground mass of the metal. A second compound, KCd_7, is formed at 5.2 per cent. K and 473° C. by a reaction between KCd_{11} and the liquid. At 468° C. the solubility curve cuts the freezing point curve of KCd_7 at 6.8 per cent. K and we have two conjugate liquid solutions. When solid therefore the alloys consist of KCd_7 and Cd.

Bismuth and potassium form two compounds, KBi_2 at 550° C. and K_3Bi at 670° C. Two other compounds, K_3Bi_2 and K_9Bi_7 (?), are formed by reaction with the liquid at about 420 and 375° C. respectively. KBi_2 and Bi form a eutectic at about 263° C. and 0.5 per cent. K. KBi_2 and K_9Bi_7 (?) form a eutectic at 340° C. and 15.7 per cent. K, while between 36.1 per cent. K and 100 per cent. K we find K_3Bi crystallizing out in a ground mass of K. K_3Bi transforms in the solid at 280° C.

Tin and potassium form 4 compounds, KSn_4, KSn_2, KSn and K_2Sn, the formulæ of the two latter being doubtful. KSn and KSn_2 melt above 800° C. and determinations difficult. KSn_4 is formed by reaction of KSn_2 with the melt at 600° C. and about 5 per cent. K, while KnS_4 transforms in the solid at about 410° C. So between 0 and 12 per cent. K the alloys consist of crystals of α KSn_4 in a decreasing ground mass of tin. Similarly K_2Sn forms from KSn and the liquid at 560° C. and about 55 per cent. K, the final ground mass being K.

Lead and potassium form 4 compounds, KPb_4 (?) KPb_2, an unknown

x, K_2Pb (?) which transforms in the solid. Between 35 and 75 at per cent. K the system consist of two conjugate liquid solutions K_2Pb(?) separating about 570° C. x reacts with liquid to form KPb_2 at 340° C. KPb_2 reacts with liquid to form KPb_4 at 298° C., while KPb_4 and Pb form a eutectic at 275° C. and about 8 at per cent. K. At the K end of the diagram we have a eutectic of K and K_2Pb about 98 at per cent. K about 4–6° below the melting point of K.

Cadmium-Sodium: Kurnakow and Kusnetzow. *Zeits. Anorg. Chem.*, 52, 173. Form a eutectic at 95.4° C., 99.26 at per cent. Na. $NaCd_2$ at 384° C., eutectic at 351° C. and 19.2 at per cent. Na. $NaCd_6$ at 363.5° C. and 14.25 at per cent. Na. Eutectic at 291° C. and 5.5 at per cent. Na.

On the Freezing Point Diagrams of the Binary Systems Platinum-Arsenic and Bismuth-Arsenic: Friedrich and Leroux. *Metallurgie*, 5, 148. Pt and Pt_2As_3 1400° + forms a eutectic at 13.2 per cent. As and 597° C. Alloys up to 28 per cent. As made.

Alloys of Bi up to 13 per cent. As were made and above 3 per cent. As show two points, the lower 267° C., the upper at 480–490° C.

The Freezing-Point Diagram for the Cobalt-Arsenic Alloys: K. Friedrich. *Metallurgie*, 5, 150. Alloys made up to 53.5 per cent. As. Cobalt 1494 with max. of 1 per cent. As in solution and Co_5As_2 form a eutectic at 30 per cent. As and 916° C. Beyond this point the curve shows a continued rise to 1179° C. (CoAs), with constant points at 1014, 959 and 926° C., showing the formation of Co_3As_2, Co_2As and Co_5As_2 respectively. At lower temperatures we have changes in the solid, α into β Co_5As_2 at 828, etc.

The Transformations of Sulphur-Iron: Rinne and Boeke. *Zeits. anorg. Chem.*, 53, 338. Transformation temperature rises from 90–98° C. at 5 per cent. Fe, 95 per cent. FeS to 138° C. with 7 per cent. Fe and then remains constant, with the appearance of α iron amongst the mischkrystalle (solid solution).

Cobalt-Sulphur Diagram: K. Friedrich. *Metallurgie*, 5, 212. Cobalt and Co_4S_3 935° C. form a eutectic at 26.6 per cent. S and 879° C. From Co_4S_3 the curve rises towards a max. (CoS?). At lower temperatures we find changes in the solid, 780° C. ± and Co_4S_3 breaks down.

The Freezing Point Diagram for Nickel-Sulphur: K. Bornemann. *Metallurgie*, 5, 13. The freezing point curve falls from pure Ni at 1448° C. to 644° C. at 21.5 per cent. S, with a critical point at 1345° C. and 2½ per cent. S showing the reaction between α mixed crystals 1.8 per cent. S and the melt to form β with 2 per cent. S. From 644° C. and 21.5 S, the eutectic point, the curve rises to 787° C. (Ni_3S_2) and γ mixed crystals freeze out. At a lower temperature solid solutions break down into Ni and Ni_3S_2, the eutectoid point being 532° C. and 24½ per cent. S. Between Ni_3S_2 and Ni_6S_5 we have a series of solid solutions with changes in the solid between 550 and 445° C., the final product being a mixture of Ni_3S_2 and Ni_6S_5.

On the Sulphides PbS, Cu_2S, Ag_2S and FeS: K. Friedrich. *Metallurgie*, 5, 23, 50. PbS 1120°: Cu_2S 1135°: Ag_2S 812°: FeS 1171° C. Freezing and cooling curves reproduced.

On the Constitution of Nickel-Mattes: K. Bornemann. *Metallurgie*, 5, 61. The system FeS — Ni_3S_2 worked out. Curve falls from FeS 1194° C. to 880° C. at $(FeS)_2.Ni_3S_2$ giving FeS in a ground mass of $(FeS)_2.Ni_3S_2$. From this point to Ni_3S_2 the curve falls to 788° C. and shows a series of solid solutions which at a lower temperature rearrange themselves in the solid, the eutectiod point being 89 per cent. Ni_3S_2 and 510° C.

The system Ni_2S and FeS shows a maximum at $(FeS.)_2Ni_2S$. 886° C., which with FeS forms a eutectic at 872° C. and 54 per. cent Fes. At 575° C. it breaks down into $(FeS)_3(Ni_2S)_2$, and at 420 the latter forms $(FeS)_4Ni_2S$.

Between $(FeS)_2Ni_2S$ and Ni_2S the curve falls and we have two series of solid solutions with a reaction point at 819° C. and 33 per cent. FeS, the saturated mixed crystals containing 40 and 44 per cent. FeS. At a lower temperature there break down into $(FeS)_3(Ni_2S)_2$ 575° C. and Ni_3S 545° C., the eutectoid being at 23 per cent. FeS and 480° C.

Zinc Blende in Mattes: K. Friedrich. *Metallurgie*, 5, 114. PbS and ZnS (1625° C.?) form a eutectic at 7 per cent. ZnS and 1045° C.

Cu_2S and ZnS (1690° C.?) show a eutectic point at 1087° (to 1169° C.) which appears with 1 per cent. ZnS.

ZnS and Ag_2S form a eutectic at 800° C. and 3 per cent. ZnS.

ZnS and FeS form a eutectic at 5 per cent. FeS and 1180° C. when made with Magnetkies but at 5t per cent. FeS and 1160° C. when made from FeS 99 per cent. + Fe 1 per cent.

The Freezing-Point Diagram of the Binary Systems, Silver-Sulphide Copper-Sulphide and Lead-Sulphide Copper-Sulphide: K. Friedrich. *Metallurgie*, 4, 671. Cu_2S 1121° C. and Ag_2S 835° C. form a series of solid solutions, the freezing point curve showing a minimum between 30 and 40 per cent Cu_2S.

Cu_2S and PbS 1114° C. form a eutectic at 540° C. and 51 per cent. Cu_2S.

Alloys of Potassium Chloride with Silver Chloride and Potassium Chromate: S. Żemczużny. *Zeits. Anorg. Chem.*, 57, 267. KCl 790° C. and K_2CrO_4 984° G. form a eutectic at 658° C. and 31 mol. per cent. K_2CrO_4.

KCl and $K_2Cr_2O_7$ form a eutectic at 366° C. and 28 mol. per cent. KCl.

KCl and AgCl 451° C. form a eutectic at 306° C. and 30 mol. per cent. KCl.

Isomorphism of Potassiun and Sodium Compounds: Kurnakow and Zemczużnyj. *Zeis. anorg. Chem.*, 52, 186. NaCl 819° C. and KCl 790° C. form a series of solid solutions with a minimum at 50 per cent. (mol.) 664° C.

KBr 757 C. and NaBr 768° C. have minimum at 50 per cent. and 644° C.

NaI 660° and KI 693° C. have minimum at 55 per cent. (mol.) and 586° C.

NaFl 997° C. and KFl 837° C. form a eutectic at 40 mol. per cent. NaFl and 700° C.

The Potential and Chemical Constitution of Alloys: N. Puschin. *Zeits. anorg. Chem.*, 56, 1. Translation by J. Pinsker from *J. Russ. Chem. Gesell.*, 1907. See *Rev. de Met.*, 1907, 926.

Experimental Study of the Freezing and Melting of Cast Iron: Goerens and Gatowsky. *Metallurgie*, 5, 137. The graphite forms during the freezing of the eutectic; the longer the freezing takes the larger is the graphite and more forms. The process of graphite formation only occurs after the cementite eutectic has frozen, and therefore graphite must be looked upon as a decomposition product.

On the Identity of Graphite and Graphitic Carbon of Reheating in Cast Iron: G. Charpy. *Rev. de Met.*, 5, 75. Experiments show the two to be the same.

On the Solubility of Graphite in Iron: G. Charpy. *Rev. de Met.*, 5, 77. Some cemented Swedish bar melted down and cooled very slowly gave 3.75 per cent. total and 3.34 per cent. graphite carbon. Pieces were heated to various temperatures, 750°, 850°, 1000°, 1050°, 1080°, 1100°, 1150° C., and quenched. The combined carbon rose from 0.31 to 2.47 — with a big jump from 1.40 per cent. between 1100° C. and 1150° C. Solubility at 1000° C. is about 1 per cent.

The Solubility of Graphite in Iron: C. Benedicks. *Metallurgie*, 5, 41. Gives the views of different authorities. Taking some Swedish gray iron — C = 3.9, Si 0.75, Mn 0.37 — by rapid heating to 800° C. and slow cooling repeatedly a mixture of ferrite and graphite is produced. No trace of cementite is left. However, on reheating to about 940° C. some of the graphite is redissolved.

On the Theory of Malleableizing: F. Wüst. *Metallurgie*, 5, 7 Cementite breaks down into ferrite and temper carbon. The temper. carbon is oxidized by oxygen to CO_2 which in the iron is reduced to CO by the temper carbon. On the other hand CO reacts on the ore in which the casting is packed and CO_2 is formed once more. If there is not enough oxygen in the ore the casting may be carbonized by the CO breaking down into CO_2 and C.

Cast Iron as Cast and Heat Treated: W. H. Hatfield. *J. I. and S. I.*, 1907, II., 79. Material heated to various temperatures, 780°, 820°, 860°, 900° and air cooled. Physical tests and microstructure.

On the Influence of Phosphorus on the Iron Carbon System: F. Wüst. *Metallurgie*, 5, 73. Works out the decrease of solubility of carbon in iron, determination of melting points of iron-carbon alloy, saturated with phosphorus, fluidity of such material, transformations and structure of the solid solutions formed. 1 per cent. phosphorus lowers freezing point 27° C.; this holds good up to 6.7 per cent. phosphorus above which the freezing point is raised. A new point occurs at 970° C., independent as to temperature or phosphorus content. Intensity increases with phosphorus up to 6.7 per cent. Pearlite point is not

changed. Solubility of iron and carbon is lowered by phosphorus. A ternary eutectic is formed 6.7 per cent. phosphorus, 2.0 per cent. C, 91.3 per cent. Fe, freezes at 950° C. Above 6.7 per cent. phosphorus the phosphide Fe_3P appears.

On the Heat-Treatment of Medium-Carbon Steel. The Influence of Rate of Cooling on the Physical Properties and Structure: W. Campbell. *Metallurgie*, 4, 772. Also *Proc. A. S. Test Materials*, VII., 240. Steel of 0.4–0.5 per cent. C. shows much more ferrite and more ductility when cooled in the furnace.

The Constituents of Steels: H. le Chatelier. *Rev. de Met.*, 5, 167. Simple bodies, graphite, ferrite. Chemical compounds Fe_3C, Mn_3C. Allotropic varieties of iron. Solid solutions, austenite, martensite. The constituent *x* sorbite, troostite, osmondite, etc. Aggregates pearlite, sorbite of Osmond. Obscurity caused by inexact application of terms.

Crystallization and Structure of Steel: Baykoff. *Rev. de Met.*, 5, 177. From *An. de l'Inst. Polytech. à St. Pétersburg*, VIII., 289. Study of structure and composition of crystals found in cavities and pipes, *e. g.*, Tschernoff's crystal. Crystalline forms of manganese sulphide shown.

Applications of the Equlibrium Diagram of the Iron Carbon System to the Constitution and Thermal Treatment of Steel and Cast Iron: M. A. Portevin. *Rev. de Met.*, 5, 24. Takes up the constituents. Structure of different alloys. Thermal treatment. Remarks on cementation.

Non-Metallic Impurities in Steel: E. F. Law. *J. I. and S. I.*, 1907, 94. Study of sulphide of iron, sulphide of manganese, manganese silicate, iron silicate, oxide and their effects on steel. Excellent micrographs.

Case-Hardening of Mild Steel: Bannister and Lambert. *J. and S. I.*, 1907. Carbon probably passes into iron as a solid solution. At 871° carbon penetrated till saturation reached (0.9 per cent. C.). At 982° free Fe_3C formed on outside.

Case-Hardening: G. S. Scott. *J. I. and S. I.*, 1907, III., 120. Influence of temperature, of material, twinning resulting from ammonia treatment of bars.

Hardened Steels: P. Longmuir. *J. I. and S. I.*, 1907, III., 137. The constitution of commercially hardened steel is essentially that described by Arnold and McWilliams (1902, I., 120).

Copper Steels: P. Breuil. *J. I. and S. I.*, 1907, II., 1. Ingots 110 lbs. cast. Three series, mild, semi-mild and hard steels. Copper 0 to 32 per cent. Segregation, hardness, critical points, treatment, mechanical tests, microstructure, worked out.

Special Steels (Automobile): L. Révillon. *Rev. de Met.*, 5, 53. Reviews the different classes of steel. Cement steel 2 per cent. Ni quenched from 900° C. and tempered at various temperatures; 6 per cent. Ni also nickel-chrome steels from 900, from 780 and 750° C.

ABSTRACTS OF RECENT MINING DECISIONS.*

LOCATION OF CLAIMS.

MINING CLAIM — MARKING OF BOUNDARIES.

No particular method of marking the boundaries is required under the statute providing that a mining location "must be distinctly marked on the ground so that its boundaries can be readily traced," and what is a sufficient marking may depend on the typography of the ground. In such case, it is a question of fact whether or not the lines are so marked that they can be readily traced by a person making a reasonable effort to do so.

Charlton *v.* Kelly, (Alaska) 156 Federal 433, October, 1907.

DISCOVERY OF MINERALS.

To constitute a discovery of gold sufficient to support a location of a gold placer mining claim as against an adverse mineral locator, the gold found must be of such quantity and character and found under such circumstances as to justify a man of ordinary prudence in the expenditure of time and money in the development of the property. The word "development," used in an instruction by the court, was equivalent to the word "exploration" as required by the law.

Charlton *v.* Kelly, (Alaska) 156 Federal 433, October, 1907.

MINING CLAIM — OCCUPANCY AND POSSESSION.

The mere placing of a tent and a few tools and a small supply of provisions upon a place, mining claim, is not sufficient alone and of itself to constitute actual possession. But where a prospector has in good faith temporarily gone away from his claim for the purpose of purchasing provisions or supplies or for any other temporary purpose, intending to return and resume his actual occupation, possession, and labors, such temporay absence is not to be construed to be an abandonment of his right to the ground, and of his rights to the claim, and another entering upon the claim during such temporary absence could not initiate any right thereto.

Charlton *v.* Kelly, (Alaska) 156 Federal 433, October, 1907.

MINE CLAIMS — LOCATION OF ASPHALTUM.

A deposit of asphaltum in a lode or vein in rock in place may be entered and patented by means of lode mining claims under the statutes of the United States. But it cannot be located by means of placer claims under the Act of 1897, requiring the entry of lands containing petroleum or other mineral oils.

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* Prepared for the SCHOOL OF MINES QUARTERLY by J. W. Thompson, Attorney-at-Law, Indianapolis, Indiana.

Webb *v.* American Asphaltum Mining Co., 157 Federal 203, November, 1907.

LODE AND PLACER LOCATION.

A lode location is the location of a vein or lode in mineral bearing rock or other earthy matter in place in a fissure in rock, so that its boundaries are sharply defined by rocky walls in place. A placer location is a location of a tract of land for the mineral-bearing or other valuable deposits upon or within it, that are not found in lodes or veins in rock in place; and in a claim of a tract of land for the sake of loose deposits on or near its surface. Each must be located according to the separate provisions of the United States Statutes.

Webb *v.* American Asphaltum Mining Co., 157 Federal 203, November, 1907.

MINING CLAIM — LOCATION — METALLIFEROUS AND NONMETALLIFEROUS SUBSTANCES.

Section 2320 of the United States Statute specifies the method by which veins or lodes of quartz or other rock in place bearing gold, silver, cinnabar, lead, tin, copper, or other valuable deposits, may be entered and patented. And the terms, "all mineral deposits," or "other valuable deposits," include nonmetalliferous deposits, such as alum, asphaltum, borax, guano, diamonds, gypsum, resin, marble, mica, slate, amber, petroleum, limestone, and building stone, as well as metalliferous metal, such as gold, silver and other metals.

Webb *v.* American Asphaltum Mining Co., 157 Federal 203, November, 1907.

RELOCATION — LOCATION WORK.

The relocation of an abandoned lode claim must be made by sinking a new discovery shaft and fixing new boundaries in the same manner as if it were a location of a new claim; or, in lieu thereof, the relocator may sink an original discovery shaft ten feet deeper than it was at the time of the abandonment. No less work, and work which is not a substantial compliance with this provision, will amount to a relocation of an abandoned lode claim.

Ingemarson *v.* Coffey, (Colo.) 92 Pacific 908, December, 1907.

LOCATION OF MINING CLAIM — NOTICE — LOCATION WORK.

The sixty days within which the locator must perform his location work begins to run from the date of the discovery of mineral and posting his discovery notice; and a locator cannot extend his time to do his work by changing the date on his discovery stake, or by renewing his notice of discovery. There cannot be two discoveries of the same thing by the same person, and the law having fixed the time in which the work must be performed, the locator has no power or authority to extend or enlarge that period for his own convenience. If the discovery shaft is not completed within the legal time, it is mere folly to pull down the old notice and put up another of a later date, as the sixty

days begins to run from the date of discovery, and no self-serving act of the locator can enlarge the time. The exact rule may be stated thus: The locator is not permitted to re-enter and oust himself, predicating such re-entry and ouster on his own delinquency, and permitting him to re-enter and re-oust himself periodically to save the necessity of developing his claim. To permit the original locator the power within himself to make effectual a forfeiture arising from his own delinquency by perfecting a relocation is to place in his hands the extraordinary privilege of holding mineral lands perpetually without doing any location work whatever.

Ingemarson *v.* Coffey, (Colo.) 92 Pac. 908, December, 1907.

Failure to Comply with Mining Rules.

The United States statutes give the miners of every mining district authority to make regulations not in conflict with any laws of the United States, or of any state or territory, in which the district is situated. Under the regulations of mining districts in California which provide for posting two notices to establish a location, it is held that the failure to comply with such mining rules in this respect, would not work a forfeiture of title, in so much as there was nothing in the rule which made non-compliance a cause of forfeiture; that, unless the rule so provided, the failure to comply with its requirements would not work a forfeiture. This is the rule in California and Arizona. But the opposite is held in Montana and Nevada.

Yosemite Gold Mining & Milling Co. *v.* Emerson, 28 Sup. Ct. Rep. (U. S.) 196, January, 1908.

Relocation of Mining Claim — Failure to Perform Assessment Work — Resumption of Work.

On the failure to perform the assessment work a mining claim is open to relocation in the same manner as if no location had ever been made, provided the original locators, their heirs, assigns, or legal representatives have not resumed work on the claim after failure and before such relocation. But where there has been a failure to perform the assessment work within the time specified, but the original locator, after such failure had, in good faith, resumed work upon the claim, the claim is not then subject to relocation, on the ground of the original failure. Whether or not such work has been resumed is a question of fact; and when a trial court or jury, has so found, such finding will not be disturbed on appeal.

Yosemite Gold Mining & Milling Co. *v.* Emerson, 28 Sup. Ct. Rep. (U. S.) 196, January, 1908.

Relocation.

A mining claim properly located is not open to relocation until the rights of the former locator have come to an end. A relocator cannot avail himself of mineral in the public lands which another has discovered, until the discoverer has in law abandoned his claim and left

the property open. The right of location is given by Congress, and can only be exercised within the limits prescribed by the grant; and a location can only be made where the law permits; and any attempt to go beyond that will be unavailing. Hence, a relocation on lands actually covered at the time by another valid and subsisting location, is void not only against the prior locator, but all the world, because the law does not permit it to be done.

Nash *v.* McNamara, (Nev.) 93 Pacific 405, January, 1908.

MINING CLAIM — RELOCATION.

There can be no relocation of a mining claim while it is in the actual possession of persons who have done the requisite amount of assessment work, although under insufficient location. The rule is that for the purpose of relocation, the land must be open and subject to location.

Ware *v.* White, (Ark.) 108 Southwestern 831, January, 1908.

POSSESSION OF MINING CLAIM — PRESUMPTION.

The mere possession and improvement, in the absence of other things, will give no value to a mining claim; their only force is to raise a prima facie presumption that the possession is rightful, and thus prevent the land being subject to original location as wild and unimproved land.

Ware *v.* White, (Ark.) 108 Southwestern 831, January, 1908.

LOCATION OF CLAIM — RIGHT OF JUNIOR LOCATOR.

A junior locator, who has relocated a claim upon a senior invalid location, is entitled to hold the claim for ninety days; and by instituting an action to recover the claim before the expiration of the ninety days, he would be entitled to recover a judgment for possession to the end of such period; and if he should fail to do the required work within the ninety days, the claim would then become subject to relocation by others.

Nash *v.* McNamara, (Nev.) 93 Pacific 405, January, 1908.

LOCATION OF CLAIM — RIGHT OF POSSESSION.

The owner of a valid lode location has the exclusive right of possession and enjoyment of all the surface included within the lines of the location, and this right of possession forbids any trespass. Without the owner's consent or acquiescence, no one can rightfully enter upon the premises, or disturb its surface by sinking shafts or otherwise. It was the intention of Congress to secure the fullest working of the mines and the complete development of the mineral property, and for this purpose the owner should have the undisturbed possession of not less than a specified amount of surface; and the exclusive right of such possession is as much the property of the locator as the vein or lode by him discovered and located.

Nash *v.* McNamara, (Nev.) 93 Pacific 405, January, 1908.

MINING CLAIM — POSSESSION — TAXATION.

A statute provided that mining claims, whether patented or entered for patent or not should be assessable and taxable on account of producing gross proceeds, then and in that case the possession should be the subject of assessment, and in case such mining properties should be sold for taxes levied, such sale should pass the title and right of possession to the purchaser. Under this statute a valid subsisting mining location was property distinct from the land itself and taxable; the right of possession of the land for mining purposes, and not the land itself, was the subject of taxation; that a tax deed under a sale conveyed the right of possession, but affect no interest of the United States; the tax deed upon its face was a valid instrument, and after the statutory period of limitation constituted a superior title to the claim of a grantee from the person originally in possession.

Elder *v.* Wood, 28 Sup. Ct. Rep. 263, January, 1908.

MINING LAND — NOT TAXABLE BEFORE PATENTED.

Where a mining claim is located on the lands of the United States, it is still a part of the public domain, and although proceedings have been begun by the owner of the claim for the acquisition of the title to the land by patent, still the land itself is not subject to taxation by state authorities.

Elder *v.* Wood, 28 Sup. Ct. Rep. 263, January, 1908.

CONFLICTING RIGHTS.

MINING CLAIM — PARTNERSHIP — BREACH OF AGREEMENT — FRAUD.

Under an oral agreement one person was to locate and mark certain mining claims and prepare the notices of location; and the other parties were to have such notices duly recorded at their own expense. This agreement made the parties partners engaged in dealing and in working mining claims. The first party proceeded to locate the mining claims, prepared the necessary notices, and delivered them to the other parties. The parties as such partners proceeded under the agreement and engaged in dealing in and working mining claims. The second parties fraudulently withheld all the notices from record until after the time for recording the notices had expired; they thereupon, and after the expiration of such time, relocated all such mining claims for their own benefit. In an action by the first party to recover his interest in such claims, the court held that all such persons became partners for mining purposes from the time of the original agreement and were all bound thereby; that the action of the second parties in failing to record such notices within the time and perfect such locations, and in relocating the claims, was for the fraudulent purpose of cheating such first party out of his half interest; and that the first party was accordingly entitled to recover his interest in all such claims.

Cascaden *v.* Dunbar, 157 Federal 62, October, 1907.

MINING CLAIM — QUIET TITLE — ADVERSE CLAIM.

Where a person after filing adverse claim, sued in the state court to quiet his title, under state laws, such suit was not affected by the statutes of the United States which provide that where an adverse claim is filed, publication of notice of an application for a patent, all proceedings, except the publication of notice and the making and filing of the evidence, shall be stayed until the controversy shall have been decided by a court of competent jurisdiction, or the adverse claim waived, etc.

Bernard *v.* Parmelee, (Cal.) 92 Pacific 658, October, 1907.

CONVEYANCE OF MINERAL LANDS — ESTOPPEL.

The owner of land conveyed the same by deed without reservation as to minerals. The grantee immediately deposited his deed for record. On the following day, in the presence of the first grantee, the original grantor executed and delivered to another person a deed for the minerals under such land, and in the presence of the first grantee received the purchase price of such minerals. Under such circumstances the original grantee was estopped to claim any interest in the minerals. While the purchaser of the minerals had constructive notice of the first deed, his conduct in paying the consideration for the minerals was sufficient to inform the first grantee that he had no actual notice of the original conveyance, or he would not have paid for the minerals knowing that the land had been conveyed to another. The conduct of the original grantee operated as a fraud upon the grantee of the minerals.

Virginia, etc., Co. *v.* Campbell (Ky.) 105 Southwestern, 129, November, 1907.

RIGHTS AND LIABILITIES OF JOINT OWNERS OF MINES.

The extraction of coal by one tenant in common of a mine, without the consent of the other, is regarded as waste, for which he is liable to account. He will not be heard to say that his co-tenant may proceed to take coal from other parts of the vein or mine owned in common. While the coal in land is partible, still there is an appreciable element of uncertainty touching it, great enough to forbid a court of equity from depriving a party of his right in what is in the land for what may never be there. Hence, one tenant in common who has taken coal from the common mine cannot compel his co-tenants to take their share in undeveloped land with its coal which they could mine if they chose, but must account to them for their share of the money received from the coal mine.

Robertson Consolidated Land Co. *v.* Paull, (W. Va.) 59 Southeastern, 1085, December, 1907.

TRESPASS — TAKING ORE — DAMAGES.

Where a person in the honest belief that he was in the lawful exercise of a right, entered upon the premises of another and removed ore, coal and other products, he could only be held liable in damages for the actual value of such ore, coal and other products in their original places.

But a person who wilfully and intentionally took ore from the property of another, was held liable to the owner for the full value of the property at the time of such taking, without any deduction for the labor or expenses incurred in removing and marketing the same. And such wrongful taking would not divest the owner of the title, and a third person purchasing such coal or ore with knowledge that it had been so wrongfully taken, was held liable to the owner for its conversion.

United States *v.* Ute Coal and Coke Co., 158 Federal 20, December, 1907.

RIGHT TO MINE COAL — CONVEYANCE OF SURFACE.

The owner of coal lands sold and conveyed the surface, expressly reserving in the deed "all the coal underlying the same, with all mining rights and privileges appurtenant thereto." Under this reservation the grantor had the right to sink a shaft upon the lands so sold for the purpose of mining the coal thus reserved. The rule in such case is that one who has the exclusive right to mine coal upon a tract of land has the right of possession even as against the owner of the soil, so far as it is necessary to carry on his mining operations. The same rule was otherwise stated thus: "An express grant of all the minerals and mining rights in a tract of land is by natural implication the grant also of the right to open and work the mines, and to occupy for those purposes as much of the surface as may be reasonably necessary."

Baker *v.* Pittsburg C. & W. R. Co., (Pa.) 68, Atlantic 1014, January, 1908.

OWNER OF SURFACE OF MINERAL LAND — RIGHT TO SUPPORT.

Where the surface soil of mineral land is owned by one person, and the mineral underneath is owned by another, the owner of the surface has an absolute right to the necessary support for his land; and if the owner of the minerals in removing them removes the support for the surface, he is liable for all damages that result from the surface soil by reason of subsidence, regardless of how carefully or skillfully he may have conducted his mining operations.

Collins *v.* Gleason Coal Co., (Iowa) 115 Northwestern 497, March, 1908.

MINERAL LANDS — SALE OF SURFACE — SUPPORT.

Where the owner of mineral lands sold the surface, reserving to himself the minerals underneath in general terms, there was held to be an implied covenant to so mine as not to materially injure the surface. Such reservation of itself gives no right to remove absolutely the necessary support to the surface. And on the other hand the right of the grantee of the surface to subjacent support was not waived, and could only be waived by express words or by the clearest implication.

Collins *v.* Gleason Coal Co., (Iowa) 115 Northwestern 497, March, 1908.

REMOVAL OF MINERALS — INJURY TO SURFACE — MEASURE OF DAMAGES.

Where the surface soil is injured by the removal of mineral without leaving the proper support, the damages must be based on the injury to the surface and improvements caused by such mining; and that such damages would be the difference between the value of the land immediately before the commencement of the mining, and such value immediately after the mineral was removed; and that in determining such difference the damage to improvements might also be considered.

Collins *v.* Gleason Coal Co., (Iowa) 115 Northwestern 497, March, 1908.

MINING CLAIM — RIGHTS OF TENANTS IN COMMON.

A tenant in common of a mining claim who incurred expenses in prospecting, without the consent of his co-tenants, was held not entitled to recover contribution from them. But where one tenant had operated a mine and was sued by his co-tenants for an accounting for the profits, the operating tenant was held to have the right to set off as against such non-operating tenants the cost of the necessary improvements, on proof that the improvements were necessary and increased the value of the common property.

Wolfe *v.* Childs, (Colo.) 94 Pacific 292, March, 1905.

MINING LEASES.

MINING LEASE — FAILURE TO BEGIN WORK — CANCELLATION.

In mining leases, giving the lessor a share of the product or a tonnage of ore or other product, even where there is no covenant to begin work within a certain time, there is an implied covenant to begin in a reasonable time, because the consideration in contemplation of both parties otherwise fails. This rule prevails where no time is fixed in the lease for the work to begin, where it appears from the lease that time was the essence of the contract; and under the further rule that in such contracts where no time is expressed, the law implies a promise to perform within a reasonable time.

Starn *v.* Huffman, (W. Va.) 59 Southeastern 179, October, 1907.

OIL AND GAS LEASE — NATURE AND CONSTRUCTION.

An oil and gas lease conveys no present vested interest in the oil and gas in place. The interest conveyed by such a lease is held to be a mere license to explore, an incorporeal hereditament, a profit, a prendre.

Phillips *v.* Springfield Crude Oil Co., (Kans.) 92 Pacific 1119, December, 1907.

OIL AND GAS LEASE — OPTION — FORFEITURE.

An oil and gas lease gave the lessee the exclusive right for a term of years to enter upon lands and prospect for oil and gas; it also provided that in case no well was drilled within six months all rights and obliga-

tions under the lease should cease, unless the lessee elected to continue the lease by payment in advance of an annual rent of $1.00 per acre for all the land; the lease contained no covenant or agreement on the lessee's part to drill a well or pay the rental. This was held a mere naked option, and the failure of the lessee to drill a well within the time or pay the rentals, forfeited the lease.

O'Neill *v.* Risinger, (Kans.) 93 Pacific 340, January, 1908.

COAL LANDS.

Coal Lands — Entry and Purchase.

There is nothing in the coal land statute intended to prevent either individuals or corporations from acquiring any amount of coal land which it may be possible to pay for or secure. It is only meant that neither an individual nor corporation, either by direct entry in the grantee's own name, or indirectly by procuring an entry by an agent or servant, shall take from the government more than the limited quantity. An individual having exercised the right to entry may buy of others by ordinary purchase as much more as he pleases; but he cannot acquire another acre from the government. If he does so either by direct application in his own name or by employing another to make an entry and then convey to him, he thereby becomes involved in conspiracy, perjury and fraud. The statute prohibits the right of preëmption but not of the title to the land, though it may be but equitable.

Pereles *v.* Weil, 157 Federal 419.

Coal Land — Entry — Violation of Department Regulations.

The violation of regulations for entering coal lands cannot be made the subject of a criminal prosecution in the absence of a statute distinctly declaring such violation to be a criminal offense.

United States *v.* Keitel, 157 Federal, 396, December, 1907.

Coal Lands — Indictment for Illegal Entry.

An indictment which charged that the defendants conspired to hire others to make entry and purchase of public lands of the United States under the coal land act, with money furnished by such defendants, the lands to be conveyed to a corporation to enable it to acquire a larger quantity of such land than it could otherwise legally acquire, did not charge a conspiracy to defraud the government, where it was not averred that such entrymen were not legally qualified to make such purchases, nor that they severally acquired more land than the acreage limited by the act.

United States *v.* Keitel, 157 Federal, 396, December, 1907.

LIABILITY OF INJURY TO MINER.

Miners — Fellow Servants.

A ground foreman in mining operations, conducted under the supervision of a general superintendent or manager, is a fellow servant with

the miners whose operations he is directing, especially so where his duties were to direct the miners where to set up the drill, direct the shovelers where to shovel, direct the location of the underground tracks, and look after everything under ground; and where it was a common practice for him at times to help the workmen who were laboring under him in actual drilling of holes, laying the track, and other such work, and where he was subject to the general orders of the superintendent, and notwithstanding he hired men to work in his department and discharge them, but did not pay their wages.

United Zinc Companies *v.* Wright, 156 Federal 571, October, 1907.

INJURY TO PERSON EMPLOYED IN A MINE — CONTINUOUS EMPLOYMENT.

The business of an employee in a mine was to work in a blacksmith shop on one side of a tram railway. The place where the drinking water for the miners was kept was on the opposite side of such railway and it was customary for the employees to go to such place for water. Such employee in crossing the tram railway while returning from getting a drink of water was injured by reason of negligence attributable to the mine owner in operating cars upon such tram railway. The court held that at the time of his injury such employee was entitled to the same immunity from injury from negligence as when actually at work in the shop.

Woodward Iron Co. *v.* Curl, (Ala.) 44 Southern, 969, November, 1907.

INJURY TO MINER — SUFFICIENCY OF APPLIANCES.

Where a miner was injured by falling timbers caused by a defective chute, an instruction that it was the mine owner's duty to provide the miner with a reasonably safe and sufficient rope and chute to hoist timbers from a mine, to make a reasonable inspection thereof, to keep the same reasonably safe, and to use ordinary care in so doing was objectionable as placing too great a burden upon the mine owner and the correct rule was held to be that the mine owner was only required to exercise ordinary care to furnish reasonably safe appliances and to keep them in reasonably safe condition.

Leary *v.* Anaconda Copper Min. Co., (Mont.) 92 Pacific 477, November, 1907.

INJURY TO MINER — RISKS ASSUMED.

A miner injured by timber falling back upon him by reason of a defective chute, was held to have assumed the ordinary risks of the business, but not any extraordinary risks, unless it appeared that he was aware of such at the time of his employment, or that, upon learning of their existence, he continued in the employment after the lapse of a reasonable time for the defects to be remedied or removed.

Leary *v.* Anaconda Copper Min. Co., (Mont.) 92 Pacific 477, November, 1907.

INJURY TO MINER — DAMAGES NOT EXCESSIVE.

A miner twenty-four years of age and earning $4.00 a day, was injured by the fall of rock from the roof of an entry of a mine. The

injury consisted of a multiple fracture of both bones of one leg; as a result, the leg was shortened, weakened, stiff, and there was also a loss of motion. Physicians testified that the injury was permanent and would probably result in a club foot. Under such circumstances a verdict for $5,000 was not excessive.

Garard *v.* Manufacturers' Coal & Coke Co., (Mo.) 105 Southwestern 767, November, 1907.

INJURY TO MINER — INSTRUCTION AS TO DUTY OF MINE OWNER.

In an action by a miner for injuries from rock falling from the roof of an entry, an instruction substantially as follows, was held correct: It was defendant's duty to make and keep the entry in its mine, where plaintiff would have to pass, reasonably safe from the falling of overhanging rock, earth and débris in the roof of said entry, and it was not plaintiff's duty to examine said roof to see that it was safe, but he had a right to rely upon the defendant to see that said entry was kept reasonably safe, and that it had exercised all reasonable care and diligence in that regard. If, therefore, you believe from the evidence the defendant or its agents in charge of said mine knew, or by the exercise of reasonable diligence, would have known, that the roof of an entry in its mine, where plaintiff would have to pass, was unsafe from overhanging rock, earth or débris, and negligently failed to remove such earth, rock or débris, or to make the same reasonably secure, and plaintiff while in discharge of his duty, and in the exercise of reasonable care and caution on his part, was injured by the fall of overhanging rock in the roof of such entry, then your verdict shall be for the plaintiff.

Garard *v.* Manufacturers' Coal & Coke Co., (Mo.) 105 Southwestern 767, November, 1907.

INJURY TO MINER — EXPLOSION OF DYNAMITE.

A miner of eight years experience, and who had been "head man" in drill work in a mine for over a year, was guilty of such contributory negligence as to prevent recovery for injuries from an explosion of dynamite, where it appeared that he had knowledge that the bar with which he pushed and tamped the dynamite into its place was defective and where he determined for himself the depth of the holes and how to put in the sticks off dynamite and used his own judgment as to the force necessary to get such sticks in place, and where he admitted that he was pushing hard on the dynamite at the time of the explosion, and he knew that pushing too hard would explode it.

Thornberry *v.* Old Judge Min. Co., (Mo.) 105 Southwestern 659, December, 1907.

INJURY TO MINER — LIABILITY OF MINE OWNER.

The Miner's Statute of Missouri requires entry ways in mines to be run parallel for ventilation purposes, and that there should be cross-cuts not more than fifty feet apart. The statute also requires that all shots prepared by the miners for extracting coal from the solid shall be so

placed, drilled and charged that when fired they shall perform safely the work required of the shots. And where two entries were allowed to converge until only a few feet apart at a point where by law a new cross-cut should be made, and a shot was driven deep into the dividing wall, and when fired it broke into the other entry and killed a miner; it was such a violation of the statute as made the mine owner liable.

Kirby *v.* Manufacturers Coal & Coke Co., (Mo. App.) 106 Southwestern 1069, November, 1907.

MINING COAL — KNOWLEDGE OF DANGER.

Where a miner of coal was ordered to do certain mining work in the most hazardous way, it was held to afford no ground for recovery for injury, where it appeared that the miner knew of the danger incident to the doing of the work, in that particular way, and with such knowledge continued the work.

Kirby *v.* Hillside Coal Co., (Ky.) 106 Southwestern 278, December, 1907.

DUTY OF SHOT FIRER — CONTRIBUTORY NEGLIGENCE.

The Statute of Missouri requiring the shot firer, on discovering that a drill hole is dripping too much, or that it is drilled too much into the tight, so that it may, in his judgment, prove dangerous, to pass it without firing and promptly notify the mine foreman, only imposes diligence and caution on the shot firer, and does not necessarily make him negligent in failing to discover the danger in a shot fired by him.

Kirby *v.* Manufacturers Coal and Coke Co., (Mo. App.) 106 South western 1069, December, 1907.

INJURY TO MINER — PROXIMATE CAUSE.

A miner was overcome by gas in a mine where he was working, and was found in an unconscious condition. The parties who found him laid him on the cage, two sides of which were enclosed and two open, and by reason of the projection of one of his feet beyond the open side of the elevator, his foot was caught by the timbers on the shaft and his leg broken. Conceding that the mine owner was negligent in permitting a dangerous quantity of gas in the mine, yet this was said to be the remote and not the proximate cause of the injury to the miner's leg; that the proximate cause of such injury was the negligence of those who placed him on the elevator, and that the owner of the mine was not liable.

Teis *v.* Smuggler Mining Co., (Colo.) 158 Federal 260, December, 1907.

VENTILATION OF MINE — MINER MAY RELY ON EXERCISE OF CARE.

As it is the duty of the mine owner to use every reasonable precaution to have his mine in a reasonably safe condition for the miners, a miner working in such mine may rely upon the implied assurance and superior knowledge of his employer that the premises are reasonably safe for the purposes for which they are being used, unless a danger in-

cident to the employment is known to the servant, or the danger is otherwise obvious to him.

Black Diamond Coal and Mining Co. *v.* Price, (Ky.) 108 Southwestern 345, January, 1908.

VENTILATION OF MINES — DUTY OF OWNER.

A mining company, or the owner of a mine, is under duty to remove the cause of bad ventilation in its mine; and this duty is the same whether such bad ventilation is due to a defect in the general plan of ventilation in use, or to the insufficiency of the apparatus for distributing fresh air. It is the duty of the mine owner to take every reasonable precaution to have his mine in a reasonably safe condition for the performance of the duties of the miners.

Black Diamond Coal & Mining Co. *v.* Price (Ky.) 108 Southwestern 345, January, 1908.

INJURY TO MINER — NEGLIGENCE — FAILURE TO OBEY STATUTE.

A statute required coal mine operators to have the roadways in their mines sprinkled, and required the inspectors to see that all possible precautions were taken against the occurrence of explosions which might be caused or aggravated by the presence of dust. In an action by a miner for injuries it was held that the wilful failure of the manager or examiner to observe the statute was the wilful failure of the operator, though the operator had in fact no actual knowledge of the delinquency of the manager or examiner, and though the examiner had a statutory certificate. And the manager's knowledge that for many days the roadway in the mine needed cleaning or sprinkling, and failed to have it done, was a conscious omission to perform a duty and a wilful violation of the statute.

Davis *v.* Illinois Collieries Co. (Ill.) 83 Northeastern 836, February, 1908.

INJURY TO MINER — SCOPE OF EMPLOYMENT.

A miner was employed to drive an entry in a mine. He was injured while engaged in assisting others to set a prop at a place where the entry had been completed, and where it was the duty of the owner of the mine to protect the roof of the entry; he had received no directions or instructions to set the props or to protect the roof of the entry. But the mine owner had failed to perform his duty in this respect, although he had promised to do so. The miner was in the entry at his proper place and performing his work, when the necessity arose to try to protect himself and others against certain slate which was about to fall, and which did fall, between them and the mouth of the entry of the mine. The miners could not know in advance the extent to which the slate would fall, whether it would cut them off from leaving the mine or not, and in addition to this they could not perform their labor and run the cars into and out from the slate which was knocked down from the roof, if this slate was permitted to fall. Under such circumstances the miner was within the scope of his employment; the propping of the roof was

a thing necessary to be done for the protection of the miners and of the property of their employer.

Ballon *v.* Potter (Ky.), 106 Southwestern 1178, January, 1908.

INJURY TO MINER — BLASTING WITH DYNAMITE.

A miner who was familiar with the ordinary dangers of blasting with dynamite, was not entitled to recover for injuries occasioned by a delayed explosion, where he determined for himself the length of time he should wait, and where he knew there was a possibility that a charge which had not exploded might explode later.

Wilson *v.* Atlantic Crused Coke Co., (Pa.) 68 Atlantic 671, January, 1908.

INJURY TO MINER — CONTRIBUTORY NEGLIGENCE.

Where a coal miner fired a blast in an unskillful and unlawful manner which caused an explosion of gas that was generated by a former shot, it was held not to preclude a recovery by him from the owner or operator of the mine for injuries caused by an aggravation of the explosion resulting from the air being charged with dust, if such explosion would not have injured him except for such aggravation.

Davis *v.* Illinois Collieries Co., (Ill.) 83 Northeastern 836, February, 1908.

MASTER AND SERVANT — INJURY FROM BLASTING.

A master is under no duty to warn or instruct a servant as to dangers discoverable by the exercise of ordinary care. And a servant engaged in blasting should know or ought to have known the danger in charging a hole with powder within a few minutes after springing the hole, and he was not entitled as of right to any warning on the part of the master, and that under such circumstances the master was not guilty of negligence in failing to instruct or warn the servant as to the danger incident to the work.

Hardy *v.* Chicago, etc., R. Co., (Iowa) 115 Northwestern 8, February, 1908.

INJURY TO SERVANT BY PREMATURE EXPLOSION — DEFECTIVE FUSE.

The fact that a particular brand of fuse furnished by a mining company for use by its miners, was in general use for such purposes and was favorably known, was not conclusive evidence that the company was not guilty of negligence in furnishing that particular kind of fuse to its miners and the fact that a better and safer kind of fuse had been introduced within recent years and come into common use, was not conclusive evidence that the company was negligent in using the older kind. Such facts were said to be evidence for the consideration of the jury in determining whether the employer failed to exercise the degree of care required by the circumstances, the nature of the instrumentalities and the danger to be anticipated from defective appliances.

Wiita *v.* Interstate Iron Co., (Minn.) 115 Northwestern 169, February, 1908.

INJURY TO MINERS — FALL OF ROOF — NEGLIGENCE.

A miner was injured by the fall of the roof of an entry of a coal mine in which he was working. The evidence tended to show that there had been a previous fall of rock from the roof of this entry at or near the place of the accident during the previous day, and another at the place of the accident two or three days before, and that on the day of the accident the roof was badly cracked and broken and had openings in it three or four inches wide. This condition of the roof was known to the agents of the owner and operator of the mine, whose duty it was to look after the roof and mark dangerous places. The mine examiner, after the last fall of rock on the night before the accident, examined the place but put up no danger signals. There was some contradictory evidence as to the safety of the roof at the point in question, just prior to the accident and that it did not need support, and that the owner and operator of the mine did not know of its dangerous condition; and that it had performed all duties required of it by the law for the safety of the miners. Under such evidence it was the duty of the court to submit the question of the owner's negligence as a question of fact to be determined by the jury.

McCarthy *v.* Springs Val. Coal Co., (Ill.) 83 Northeastern 957, February, 1908.

INJURY TO MINER — CONTRIBUTORY NEGLIGENCE.

A miner while driving a mule and car along an entry in a mine was injured by the fall of the roof of the entry. He had never worked in the entry before and had no knowledge of any previous fall of rock. The only light was a lamp on the front of his cap. It appeared that as he passed along the entry he examined the roof of the entry for indications of loose rock, but found none. Under the circumstances it was impossible for him to make a minute examination of the roof of such entry but relied on the inspection which other employes engaged for that purpose were supposed to make. Under such circumstances the miner was held not guilty of contributory negligence.

McCarthy *v.* Spring Val. Coal Co., (Ill.) 83 Northeastern 957, February, 1908.

MINER NOT BOUND TO INSPECT.

Where the dangerous condition of the entry of a mine was known to the operator of the mine, it was held that a miner who was injured by a fall of rock from the side of an entry, without any notice or knowledge of its dangerous condition, was not required to examine or inspect the entry closely in order to ascertain its condition.

Smith *v.* Garrison, (Ky.) 108 Southwestern 293, March, 1908.

DEATH OF MINER — PROOF OF PREVIOUS ACCIDENTS.

In an action for damages for the death of a miner killed by falling rock in the manway of a mine, it was held proper to prove previous accidents occurring shortly before the one in question, for the purpose

of showing that the owner of the mine knew that the existing conditions in such manway were dangerous.

Hotchkiss Mt. & Reduction Co. *v.* Bruner, (Colo.) 94 Pacific 33, March, 1908.

MINER — DUTY TO USE CARE.

A miner cannot recklessly expose himself to danger and recover for injury that might have been avoided by the exercise of ordinary care upon his part, or by declining to undertake the employment with knowledge of the dangers attendant upon it. But this does not apply where the mine owner assures the miner that there is no danger, or if being present he directly orders him to undertake the work, or if the safety of the employment or appliances is doubtful and the master assures the servant that he will remedy the defects.

Tanner *v.* Wickliffe Coal Co., (Ky.) 108 Southwestern 351, March, 1908.

KNOWLEDGE OF DANGER — ASSUMPTION OF RISK.

A miner who is experienced, and knows and appreciates the dangers of an employment, and without asking assurance of safety from the promise of improvement, and undertakes the labor in the face of the known danger, he must be held to have assumed the risk, and the master will not be liable for injuries that happen to him. While he is not required to test the safety and soundness of the material or appliances furnished him, and the duty of inspection is not cast upon him, still if he knows of the weakness or unfitness of the materials or appliances, and that their use is dangerous, and that they render the place unsafe, and nevertheless uses them, he does so at his own risk, unless he is induced to continue the work upon the promise of his employer to remedy the evil, and then he can continue his work for a reasonable time only, if not repaired.

Tanner *v.* Wickliffe Coal Co., (Ky.) 108 Southwestern 351, March, 1908.

MISCELLANEOUS.

DEED FOR MINING CLAIM — PRESUMPTIONS.

Where a deed conveyed a portion of a mining claim, describing it by a particular name, the purpose of the name was to identify the property; and the presumption of law is that the grantor intended to pass the best title he had in the premises.

Collins *v.* McKay, (Mont.) 92 Pacific 295, November, 1907.

CONVEYANCE OF MINERALS — CONSTRUCTION OF DEED.

A deed conveying all the coal, ores and other minerals and metals in, under, and upon the land described was held to mean all substances in the earth's crust, sought for and removed for the substance itself, and was not limited to metallic substances, but included salt, coal, clay, stone, shale, etc.

McCombs *v.* Stephens, (Ala.) 44 Southern, 867, November, 1907.

MINING PARTNERSHIP.

To constitute a mining partnership it is necessary that the joint owners of the mine engage in working the mine or in the business of operating the mine. The mere co-tenancy of two or more persons in a mining scheme is not of itself sufficient to constitute them mining partners. But it is not essential that all the tenants join in the work of mining in order to establish the partnership; that relation may be established among such of the joint owners as actually engaged in the working or mining operations. The joint owners not so joining in the work are left to their rights and chargeable by their duties as co-tenants only.

Medar *v.* Norman, (Idaho) 92 Pacific 572, November, 1907.

SALE OF COAL — INFERIOR QUALITY — MEASURE OF DAMAGES.

Where a quantity of coal warranted to be of a certain grade was sold, but by mistake, coal of an inferior grade was delivered, this did not relieve the seller from liability for damages and the measure of damages was the difference in the prices of the different grades of coal.

Missouri & Illinois Coal Co., *v.* Consolidated Coal Co., of St. Louis, (Mo.) 105 Southwestern 682, December, 1907.

REFUSE FROM MINE — OBSTRUCTION OF STREAMS.

That the owner of a hydraulic mine used all reasonable precautions to prevent injury to others or the obstruction of navigable streams from the discharge of mining débris into them, and to ascertain the means best adapted to that end, it was held that the law could not declare that the observance of such means or precautions should exonerate the mine owner from liability where injury was actually caused.

Sutter County *v.* Nicols, (Cal.) 93 Pacific 872, January, 1908.

MINING PARTNERSHIP — GRUB-STAKE CONTRACT.

In order that a "grub-stake" contract, by which one person agrees to furnish supplies for a prospector and share in any mining claims he may discover, shall constitute a partnership relation, it is necessary that the agreement extend beyond the mere furnishing of supplies and should be in consideration of a participation in any discovery made.

Costello *v.* Scott, (Nev.) 93 Pacific 1, January, 1908.

MINING — EMINENT DOMAIN.

The exercise of the power of eminent domain cannot be invoked in aid of the business of mining for the benefit of the owner, for this is a purely private enterprise.

Sutter County *v.* Nicols, (Cal.) 93 Pacific 872, January, 1908.

MINING CLAIM — PROPERTY.

A mining claim perfected under the law is property in the highest sense of that term.

Nash *v.* McNamara, (Nev.) 93 Pacific 405, January, 1905.

MINING CLAIM — POSSESSION WITHOUT PATENT.

The right to a patent for a mining claim depends upon making the proper location, or upon having held possession during the period of the statute of limitation. A patent is a final grant of the legal title from the government for the claim previously acquired and avoids the necessity of thereafter doing the annual work. But so long as the statutory amount is expended each year upon the claim, the owner's right to exclusive possession and to extract and exhaust the ore is as complete as if he held a patent, but for which he may never apply unless he desires. The making of the location and the time for making it have no reference to the statute regulating the proceedings upon application for a patent.

Nash *v.* McNamara, (Nev.) 93 Pacific 405, January, 1908.

COAL LANDS — TAXATION.

A tract of coal land of more than 1,000 acres was leased for 100 years, a period supposed to be necessary under present conditions to mine and market the coal. Under existing circumstances it was not practicable to mine more than 85 acres annually, or 170 acres dur ng an assessment period of two years. Under the assessment laws of Virginia that amount only should have been assessed for taxes as "improved and under development."

Commonwealth *v.* Pocahontas Coal and Coke Co., (Va.) 60 Southeastern 84, January, 1908.

BOOK REVIEWS.

Lead Refining by Electrolysis. By ANSON GARDNER BETTS. John Wiley and Sons, New York, 1908. 8vo, pp. ix + 394. $4.00.

The book contains information on experimental and practical work and as the author states in the preface is for the benefit of those interested in lead refining. It is arranged in eleven chapters and three appendices of which the principal headings are as follows.

Electrolytes for lead refining, fifty-seven pages embracing amongst other topics the historical development of electrolytic lead refining; the work of Keith, Tommasi, Glaser, Betts; the production of compact deposits and prevention of crystallization by gelatine; various solutions used as electrolytes — fluosilicic acid, phenol-sulphonic acid, ethyl sulphuric acid, dithionic acid; lead fluosilicate solution and its behavior.

Chemistry of slime treatment, seventy-eight pages devoted to the methods of treatment.

Deposition of antimony from fluoride solutions, eleven pages, anodes, efficiency, impurities, costs.

Electrolytic refining of doré bullion, twenty-five pages on the Dietzel process; Moebius and Balbach apparatus; Moebius and Nebel process; an account of the installation of the Compania Minera, Fundidora y Afinadora, Monterey, Mex.

Manufacture of hydrofluoric and fluosilicic acids, six pages, experimental and large scale work; method of analysis of acid; conversion of hydrofluoric into fluosilicic acid by sand or quartz.

Choice of constants, nineteen pages taking up the comparison of series and multiple systems, costs, arrangement of plant and methods of slime treatment.

Refinery construction and operation, with refinery costs. Eighty-seven pages, ending with comparative costs of the Parkes and Betts processes and cost of electrolytic refinery.

Products, analyses from the Consolidated Mining and Smelting Co. of Canada, Trail, B. C.; from the United States Metal Refining Co. at Grasselli, Ind.; from experimental work at Troy, N. Y., seven pages base bullion, refined lead, slime.

Treatment of lead containing by-products, four pages — copper lead, hard lead, gold lead bullion.

Analytical methods and experimental work, fourteen pages — for slime, doré, refined lead, slag from fusing slime, electrolyte, etc.

The last chapter is a bibliography of thirty-three titles, more than half of which are patent specifications references.

Of the three appendices, the first deals with the plant of the Consolidated Mining and Smelting Co. of Canada, at Trail, B. C., and the methods used there (thirty-one pages). The second of twelve pages refers to the U. S. Metal Refining Co.'s plant at Grasselli, Ind. The

last deals with the treatment of lead refinery slime with solution of ferric fluosilicate and hydrofluoric acid — twenty-eight pages — solution of fresh slime in ferric fluosilicate and hydrofluoric acid solution followed by cementing out of copper, antimony and lead-arsenic.

Whilst each page is written in a clear and simple style, on first reading one gets the idea that in places the arrangement is unsystematic, as in chapter one. In the second chapter the account of the various methods of treating the slime — amalgamation; distillation, dry chlorination and fractionally distilling the chlorides produced; direct fusion with soda in a reverberatory to doré bullion as used at the Trail refinery, and various other methods — is very clearly written. The chapter on refinery construction is well worth the reading, giving a good description of the most important details of plant and operation. The comparative costs of refining by the Parkes and Betts processes gives $3.67 for the former as against $3.22 for the latter, as net working costs on bullion with 0.7 per cent. Sb, 0.8 per cent. Cu and 75 oz. Ag with a little gold. Estimates are given for a cost of a 50-ton refinery — a rough approximation.

To sum up, the book gives a clear detailed description of experiments carried out by Mr. Betts and his associates and of plants now operating the Betts process. Whilst the general reader will prefer the matter in a more condensed form, to the technical man interested in electrolytic work the book will undoubtedly be welcome, while to those carrying on research along these lines it will afford much valuable information. Some objection may be raised to an apparently frequent use of the words "might" and "probably," but they are preferable to "will" and "undoubtedly" which are used by many authors on matters of speculation, basing their conclusions on fewer premises.

The illustrations are sharp and clear, the full-page photographs (16) are good, though one or two seem unnecessary. Paper, printing and binding are also good.

W. C.

Journeys of Observation. By T. A. RICKARD. Dewey Publishing Co., San Francisco, 1907, 8vo, pp. 255 + 130.

The author states in the preface "this book records the observations made by a traveler who happened to be a mining engineer; it is supposed to belong to the type of *voyages metallurgiques*, such as were published in days before the globe had been overrun by tourists and its distant corners rendered commonplace by the exaggerations of the daily press. To the members of my profession the comment concerning the industrial conditions, geological structure, mining methods and metallurgical practice of southwestern Colorado and a progressive part of Mexico, will have the interest that comes from observations which reflect a point of view somewhat similar to their own, while to the layman the not too severely technical descriptions, aided by beautiful photographs, will afford information of a kind rarely obtainable except in periodicals devoted to technology."

The first part entitled "Among the Mines of Mexico" is a record

of a journey from New York to Mexico together with a description of the mining industry of El Oro, Pachuca and Guanajuato as observed in October, 1905. Being the record of a journey, things are described as they were met with. Thus chapter six is entitled "The City of Mexico, First Impressions, The School of Mines, Memories of Del Rio, The Meteorites, Cortez"; while chapter seven embraces El Oro, Rich Mines, The Geology of the District, The Mexican Mine, and so forth.

The second part entitled "Across the San Juan Mountains" is an account of a ride over the mountainous mining regions of southwestern Colorado in September, 1902. Starting from Ouray the ranges were crossed to Telluride, Silverton, Lake City, Gunnison, thence to Crested Butte and back. The American Nettie, the Bachelor, the Camp Bird, Smuggler Union, Liberty Bell, and Contention mines were visited. Silverton and its early history are touched upon, also the first smelters of Colorado, the Golden Fleece, the Good Hope Mine at Vulcan, Gunnison, the Ruby Chief Vein, the Coal Mine at Floresta, Ute and Ulay mines of Lake City, Journey Back to Ouray, are some of the other sections completing the book.

On reading this book the first impression one gets is that the illustrations are wonderful and that the literary style is much above that which we so commonly meet with, that Mr. Rickard must have the artistic eye and finally that he has put down in very elegant prose a record of those things or feelings the like of which most of us have experienced — and promptly forgotten. And when we come to the end of the book we are somewhat surprised at the amount of scientific information on geology, metallurgy and mining he has artfully made us swallow so beautifully sugar-coated. For there is a vast amount of valuable information, both geological, mining and metallurgical contained, especially in the first part, the second being rather of mining and geological interests.

Mr. Rickard is to be congratulated on producing a very interesting volume, at one and the same time interesting and informing both to the professional and the layman. The book is beautifully prepared, the profusion of illustrations, clear type and excellent quality of paper make it an acquisition to any library.

W. C.

First Course in Calculus. By E. J. TOWNSEND, Professor of Mathematics in the University of Illinois, and G. A. GOODENOUGH, Associate Professor of Mechanical Engineering in the University of Illinois. New York, Henry Holt and Company, 1908. Pp. x + 466. Price, $2.50.

This book is too large by about one hundred pages to allow of the presentation of the whole of it in the time usually allotted to its subject in the best schools. The thickness of the volume is, however, partly due to the presence of a chapter dealing with ordinary differential equations — which engineers ought to know something about, an excellent table of integrals, and a good index that renders the book a convenient work of reference. It is noteworthy that the book is the joint product of a professional mathematician, who is mainly responsible for the theory,

and a professor of engineering who is largely responsible for the practical aspects of the work. The treatment is based exclusively on the rigor-allowing method of limits, while the uses of the doctrine are disclosed by concrete problems and applications about equally distributed between geometry and mechanics. The usual division into differential calculus and integral calculus is not made here. Both branches are present, but they are present together, interpenetrating and mutually supporting each other throughout. The needs of the engineer have been generously regarded, but the work was not written for him alone. The arrangement of topics is in general in the order of increasing difficulty, such topics as infinite series, exhaustion of functions, singularities of plane curves and the like, being reserved till the reader's grasp of fundamentals shall have been confirmed.

C. J. KEYSER.

Architectural Composition. By BEVERLEY ROBINSON. New York, D. Van Nostrand Co., 1908. Cloth, 8vo, pp. 226.

"An Attempt to Order and Phrase Ideas Which Hitherto Have Been Only Felt by the Instinctive Taste of Designers" is the explanatory title which Mr. Beverley Robinson gives to his book on architectural composition.

We feel that the author has been successful in his attempt.

In the opening chapters he defines simply and clearly a certain number of necessary terms as applied to forms, and then tells why some combinations of these forms in architecture are pleasing and others are not. He strictly avoids all reference to taste and style, rocks upon which so many writers on this subject have been shipwrecked, and devotes his work to the broad principles of composition in mass and detail.

For the layman who wishes to know *why* he admires certain buildings and condemns others, the book will serve as an explanation. For the draughtsman or designer it serves as a guide and stimulates to deeper thought.

While any building designed in pursuance of the principles laid down in this book would probably be a successful composition, it is not intended as a hand-book for designers to make their task easier, but as an explanation of the pleasure or displeasure we receive from various combinations of form.

The author treats a rather complex subject very clearly and simply, and we wish that every student of architecture could read and assimilate the contents of this book.

W. DELANO.

Technical Methods of Ore Analysis. Third edition. By ALBERT H. LOW. New York, John Wiley and Sons, 1908. Cloth, 8vo, pp. xii + 344. Price $3.00.

The first edition of this excellent work was reviewed for this QUARTERLY by the late Professor Miller in 1906 (Vol. XXVII., p. 261).

In this new edition the original character of the book has been maintained and it has been brought up to date by the incorporation of much new material, especially in the chapters on Aluminium, Antimony, Arsenic, Calcium, Lead, Magnesium, Nickel and Cobalt, Tin, Uranium, Vanadium, Zinc, Coal and Coke, and in the Appendix. The latter contains about forty pages of new methods or modifications, much of this material having evidently been prepared while the book was in press. In this way the author has been able to include in this standard work considerable material even more recent than could possibly have been incorporated in the body of the text.

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